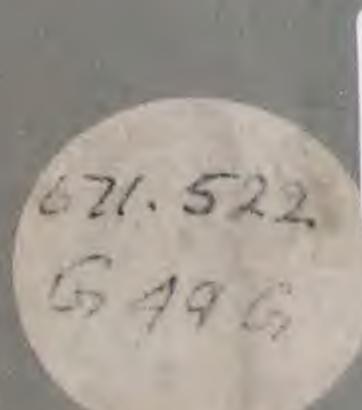# D. Glizmanenko, G. Yevseyev

GAS WELDING and CUTTING

Д. Л. Глизманенко, Г. Б. Евсеев

ГАЗОВАЯ СВАРКА
и
РЕЗКА МЕТАЛЛОВ

GAS WELDING
AND
CUTTING

by

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by B. KUZNETSOV

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This book is based in part on the course in gas welding and cutting read for welding students at the Bauman Higher Technical School (Moscow). In this edition intented for engineering colleges, the authors have also included some of the research work done by other investigators and material from other manuals and monographs on gas welding and cutting.
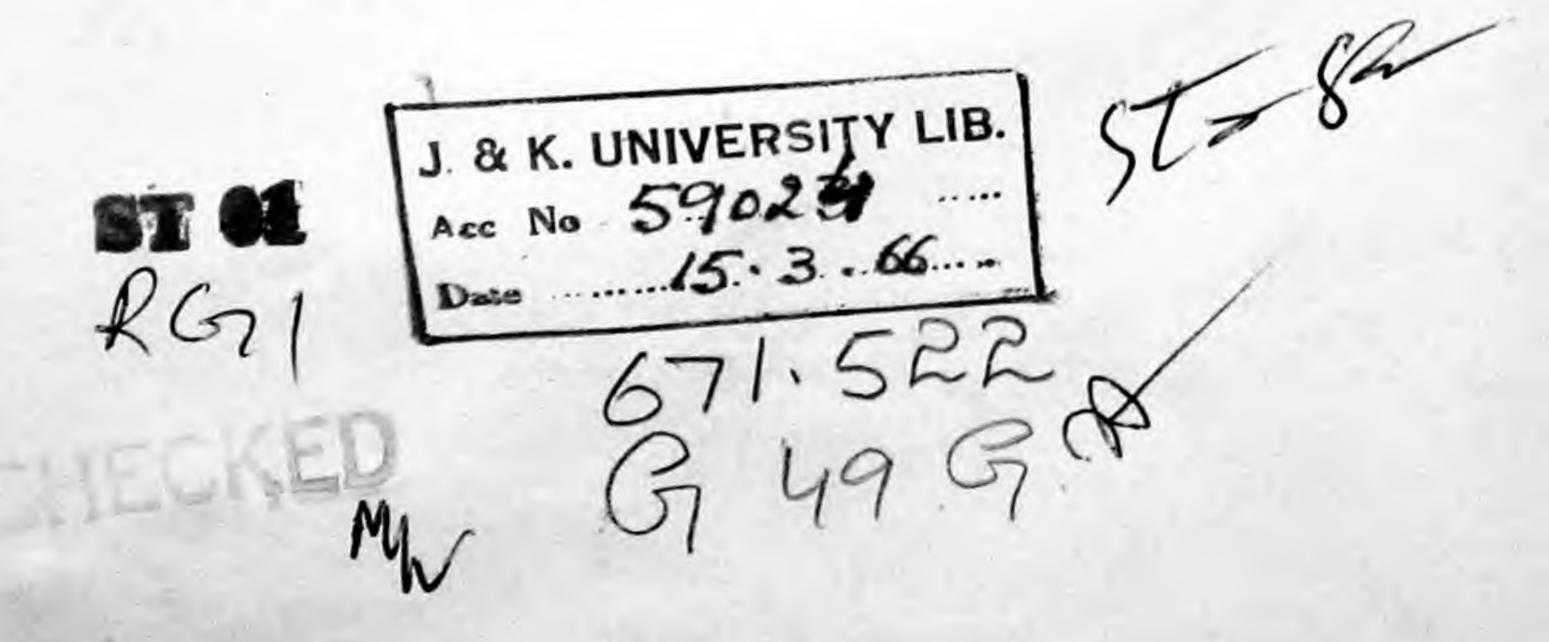
Part I of the book deals with the gases employed in flame welding and cutting, and equipment used for their production.

Part II discusses the heating of metal by the flame of fuel gases burning in a mixture with oxygen, and the flame characteristics common to all flame welding or cutting processes.

A separate part is devoted to oxy-cutting, which at present is the basic and most commonly used process among the other cutting processes. This part covers the theory, equipment and procedures of oxy-cutting.

Gas welding is likewise treated in a separate part devoted to its theory, equipment, procedures and techniques as applied to various metals and alloys.

Part IV is given over to flame hardening, soft and hard soldering, and hard-facing.

CONTENTS

PART TWO: THE OXY-GAS FLAME

Chapter VII: COMBUSTION AND THE CHEMICAL PROPERTIES
OF THE FLAME 121

**PART FOUR: TORCH SOLDERING,
HARD-FACING AND FLAME HARDENING**

PART FIVE: OXYGEN CUTTING

**PART SIX: CALCULATIONS OF STANDARD TIME
AND STANDARD OUTPUT IN
OXY-GAS WELDING. ACCIDENT PREVENTION.**

INTRODUCTION

The Soviet Union today is leading the world in the scope of welding research and its application.

In recent years, oxy-cutting has come to the fore, consuming a lion's share of all the oxygen used by the welding and cutting industry. Based on automatic machines, it has brought with it unprecedented production rates. At the same time, a commercial status has been reached by mechanised gas-pressure welding, automatic high-speed multiflame welding, flame hard-facing, torch soldering and brazing, etc.

As any of these processes involve the use of acetylene or any other fuel gas, oxygen and suitable equipment and as the metal in all of the processes is heated by an oxy-gas flame, it is customary in the Soviet Union to treat them all as a single body of flame processes. A master chart of these processes is shown in Fig. 1.

The basic trend in today's flame technology is towards the ultimate in process mechanisation, the use of special-purpose equipment of high efficiency, automatic process monitoring and control.

At the same time it should be noted that with the advent of many new electric welding methods flame welding is playing an independent part only in few fields, such as repair welding and brazing of grey, malleable and alloy cast iron; welding of copper-base alloys; hard-facing; production and repair welding of aluminium and those of its alloys which cannot yet be welded by inert-gas shielded-arc welding.

Steel is flame-welded on a limited scale, mainly where structures and pipelines from thin-walled pipes have to be erected under field conditions. The majority of such jobs is welded by manual arc welding with heavily coated electrodes, automatic submerged arc welding, inert-gas shielded-arc welding and electroslag welding. This is the reason why in the present book the procedures and techniques of flame steel welding are examined briefly.
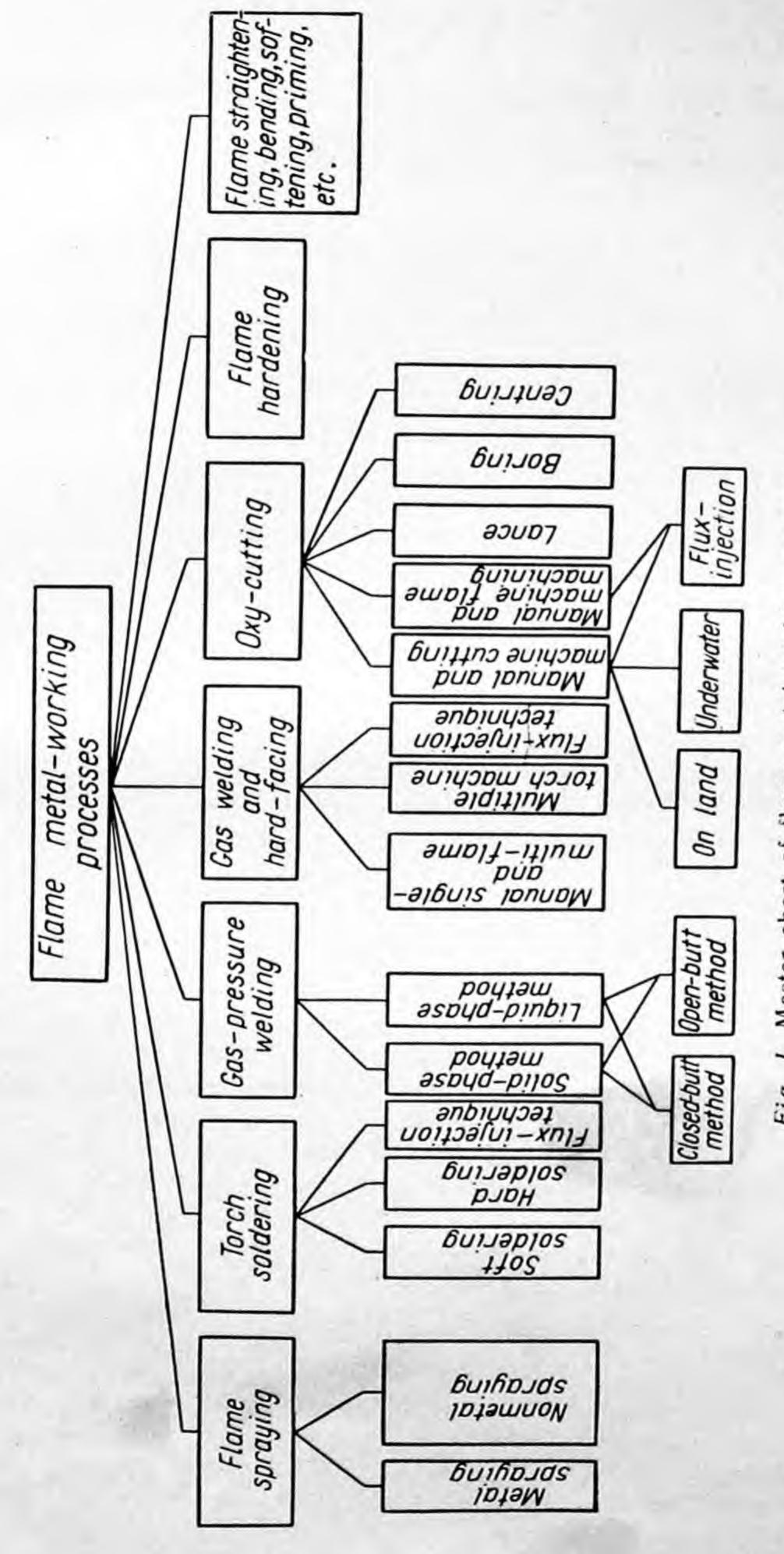
Fig. 1. Master chart of flame metal-working processes

Oxygen, fuel gases and equipment for their production, storage and transportation

CHAPTER I

OXYGEN

1. Properties

Oxygen is normally a clear gas without colour, odour or taste. For the first time it was liquefied in 1877 by reducing its temperature to below critical point and raising its pressure accordingly.

The molecular weight of oxygen is 32 and its specific gravity (air = 1) is 1.1053. The critical temperature is 154.4° K (—118.8° C), and its critical pressure 49.7 atm (gauge). Oxygen becomes liquefied at a temperature of 90.238° K (—182.962° C) at 760 mm Hg and at a temperature of 89.924° K (—183.276° C) at 735.5 mm Hg, or 1 kg/cm². Other important physical properties of molecular oxygen are given below.

Gram-molecular volume at 0° C and 760 mm Hg . . .	22.414 lit
Weight per cubic metre at 0° C and 760 mm Hg . . .	1.43 kg
Same, at 20° C and 760 mm Hg	1.33 kg
Critical volume	0.00233 m³/kg
Heat of vaporisation at 1 atm (abs)	50.92 kcal/kg
Weight of one litre of liquefied oxygen at —182.96° C and 760 mm Hg	1.14 kg
c_p (heat capacity at constant pressure) at 760 mm Hg and 0°	0.218 kcal/kg
c_v (heat capacity at constant volume)	0.156 kcal/kg
$k \left(\dfrac{c_p}{c_v} \right)$	1.397 (approx. 1.4)
Gas constant, R	26.5 kg-m/kg-deg. C
Viscosity at 0° and 1 atm (abs)	approx. 203 micropoises
Thermal conductivity at 0° and 1 atm (abs)	0.0206 kcal/(m) (h) (°C)

Oxygen is a real gas and, as such, fails to satisfy the pressure-volume relationships in exact accordance with the Ideal Gas Law, the deviation growing as the temperature of oxygen falls and its pressure rises. The behaviour of oxygen is there-

fore described by another relationship:

$$pV = KRT, \tag{I.1}$$

where K is the compressibility factor of oxygen (Fig. 2). Referring to Fig. 2, it will be seen that in the temperature range 0-25° C and the pressure range 0-300 atm (abs) K is less than unity.
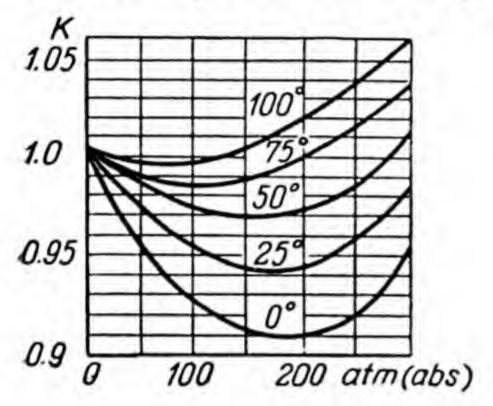
Fig. 2. Compressibility factor, K, for oxygen as a function of pressure and temperature

The reactions of oxygen with organic substances produce large quantities of heat. An increase in the temperature and pressure of oxygen appreciably speeds up the reactions. When compressed or heated oxygen is used, the velocity of reactions may under certain conditions rise continuously, as more heat is evolved by the ever greater amount of material being oxidised and as the temperature goes up. When compressed oxygen comes in contact with finely divided particles of a combustible material (coal dust, lint, etc.), mineral oils, or fats, the latter may self-ignite thus leading to a fire or an explosion. Self-ignition may in such cases be initiated by the heat given up by suddenly compressed oxygen, the heat of friction or impact of hard particles against a metal surface, an electrostatic discharge in a high-velocity stream of gas, etc.

For this reason when using compressed oxygen, care must be taken to keep it away from contact with inflammable materials.

Before use all oxygen-generating equipment and cylinders should be thoroughly degreased. In operation, measures must be taken to prevent oil or fat from getting onto oxygen cylinders and parts operating in the oxygen atmosphere. For the same reason oxygen compressor cylinders should be lubricated with pure distilled water, sometimes with an addition of 10 per cent of chemically pure glycerine, and never with oil.

Fuel gases or vapours and oxygen make explosive mixtures with various degrees of explosibility. Sometimes deflagration can undergo transition into detonation, when the resultant pressure-temperature wave advances at a rate as high as 3,000 m/sec.

Porous organic substances, such as coal, carbon black, peat, wool and fabrics, when soaked with liquid oxygen, make efficient

explosives, known as oxyliquates, which function by either deflagration or detonation. Oxyliquates are sometimes employed in mining practice.

The explosive hazard of oxygen should at all times be kept in mind when metals are flame-welded or cut with it.

2. Industrial Uses of Oxygen

For the first time the idea to use oxygen industrially was put forward by D. I. Mendeleyev of Russia in 1903 when he suggested its use for speeding up some processes, especially in metal making. Today oxygen is the basis of many industrial processes, and the field of its application is expanding.

In flame processes, oxygen is used to assist the burning of gases or vapours in order to obtain a high-temperature flame.

In metal cutting oxygen, apart from its use in the preheating flame, performs the principal function — that of burning the metal within the confined space of the kerf.

Flame processes use three grades of commercial oxygen with a purity of 99.5, 99.2 and 98.5 per cent of molecular oxygen by volume, respectively.

3. Production of Oxygen

At the present time oxygen is industrially obtained by electrolysis of water and by air liquefaction. The latter method, based on deep refrigeration and efficient fractionation, has found wider use, and any amounts of commercial oxygen may be produced from air with the least possible energy consumption per unit volume of the finished product.

The problems involved in the manufacture of oxygen may be seen from the composition of air and the boiling points of its components:

Component	Volume, %	Weight, %	Normal B. P., °C
Nitrogen (N_2) . . .	78.03	75.66	—195.8
Oxygen (O_2) . . .	20.93	23.13	—182.96
Argon	0.9325	1.2862	—185.7
Neon	0.0018	0.0012	
Helium	0.0005	0.00007	
Krypton	0.0001	0.0003	
Xenon	0.000009	0.00004	
Hydrogen (H_2) .	0.00005	0.0000036	
Carbon dioxide .	0.03	0.046	

As rough approximation, air may be regarded as a ternary mixture of 78 per cent N_2, 21 per cent O_2 and 1 per cent Ar (by volume); or as a binary mixture of 79 per cent N_2 and 21 per cent O_2. Before the components of this mechanical mixture can be separated from one another, air must be turned into a liquid state. As can be seen from the above table, the oxygen, nitrogen and argon making up liquefied air are closely boiling components. The difference between the boiling points of oxygen and nitrogen is 13° C, and it is this difference that underlies the separation of the components. When liquefied air is allowed to vaporise, the nitrogen is the first to boil off from the mixture. The mixture becomes richer in liquid oxygen as more nitrogen is evaporated. The desired purity of separation is obtained by re-circulating the mixture through the refrigerating and fractionating equipment several times. Where the primary product is pure oxygen, the argon is discharged with the nitrogen.

Air liquefaction and fractionation makes it possible to obtain practically any quantities of oxygen. The energy consumption varies from 0.45 to 1.6 kWh per cubic metre, depending on the size and set-up of oxygen plant.

The method of producing oxygen in the above manner uses special equipment and consists essentially in:

(1) removing mechanical impurities, moisture and carbon dioxide from the air;

(2) compressing the washed air to pressures from 6 to 200 atm (gauge), depending on the refrigerating plant used;

(3) reducing the temperature of the compressed air in inter-changers which use the outgoing gases, oxygen and nitrogen, for the purpose. In some types of plants, ammonia fore coolers assist in pre-cooling;

(4) reducing the pressure of the air by expansion through a nozzle (known as "throtting") to create cold and, thus, lique-faction;

(5) rectifying the liquefied air and separating the nitrogen from the oxygen.

The gaseous oxygen is then collected in a gas holder, and the liquid oxygen in a storage tank. The gaseous oxygen is compressed to a pressure of 30 atm (gauge) by a compressor when used at the site where it is produced, or is charged into cylinders where it is held under a pressure of 165 atm (gauge) when supplied to outside consumers.

The liquid from the storage tank is loaded into tank cars or trucks to be delivered to consumers.

A typical oxygen producing plant catering for the needs of gas welding and cutting in the U.S.S.R. is usually of small to medium capacity, generating 17, 30, 115 or 275 cubic metres of oxygen per hour with a purity of 99.2 to 99.5 per cent.

Table 1 presents the basic performance data on Soviet-made plant used in welding practice for commercial oxygen production. Other industries use larger types of plants with a capacity of up to 35,000 cubic metres of oxygen per hour.

Table 1

Performance of Plant Generating Gaseous Commercial Oxygen

Feature	Capacity, m³/hr			
	17	30 *	115 **	275 ***
Type designation . . .	СКАДС-17	КГН-30	УКГС-100-1	КГ-300М
Operating air pressure, atm (gauge):				
(a) starting	200	200-220	50-60	6/220
(b) steady-state . . .	100-130	110-120	30	6/100
Air flow rate, m³/hr . .	110	180	720	1,600
Installed power, kW . .	44	77	287	290
Specific energy consumption, including cylinder filling, kWh/m³	2.0	2.0	1.5	1.0-1.1
Oxygen purity, per cent	99.2-99.5	99.2-99.5	99.2-99.5	99.2-99.5
Caustic potash consumption, kg/m³ of oxygen	10-12	10-12	9-10	3-4
Floor space per plant, sq m	90	120	170	210

* Including an oxygen pump.
** Of the medium-pressure type; including an expander.
*** Of the double-pressure type.

4. Use of Liquid Oxygen in Welding and Cutting

(1) Advantages of Liquid Oxygen. When liquefied, oxygen is a clear and quick and shows a faint blue colour. It solidifies at —218.4° C and 760 mm Hg, forming crystals of blue colour. The heat capacity of liquid oxygen is 0.406 kcal/kg-deg.

Liquefied oxygen remains in the liquid state only at the end of production, during storage and in transit. Before it can be

2*

used for welding or cutting, liquid oxygen is turned to a gas by evaporation.

One litre of liquid oxygen produces 860 litres of gaseous oxygen (as measured at 20°C and 760 mm Hg); 1 kg of liquid oxygen produces 750 litres of gaseous oxygen under the same conditions.

The main advantages offered by liquid oxygen are as follows:

(a) the weight of containers for liquid oxygen is several times smaller than is required for an equivalent quantity of gaseous oxygen; requirements in transportation facilities are reduced in proportion;

(b) a smaller number of oxygen cylinders has to be kept in circulation with the accompanying reduction in cylinder maintenance costs;

(c) the pressure of liquid oxygen in containers is not higher than 1 atm (gauge), which fact enhances safety to personnel in welding shops;

(d) when it is produced at the site where it is to be used, gaseous oxygen contains little moisture, if any; this excludes condensation and ice formation in the pipeline at low ambient temperatures, and the pipeline may be left uninsulated even if it is buried in the ground above the frost line.

The disadvantages of liquid oxygen arise from the fact that losses are inevitable due to evaporation during storage, transportation and gasification. Therefore the decision to use liquid oxygen in a welding shop should be weighed against this inconvenience.

Welding shops using over 500 cubic metres of oxygen a day usually have cold gasificators evaporating liquid oxygen under a pressure of up to 15 atm (gauge) and supplying the gas over a pipeline to welding or cutting stations. In small welding shops liquid oxygen is held in a storage tank whence it is compressed by gasifying oxygen pumps into cylinders or conveyed over pipelines under the requisite pressure to welding or cutting stations.

(2) Liquid Oxygen Tanks. Tanks for liquid oxygen may be intended for either *storage* or *transportation*.

A liquid oxygen storage tank (Fig. 3) consists of a thin-walled brass sphere *1* suspended on chains *2* inside a steel shell *3*. The top of the sphere has an inspection manhole *4* closed with a lid and sealed with a solder. Outside the sphere there is a sump *7*. The space between the shell and sphere is filled with thermal

insulation consisting of powdered magnesium carbonate, mypore or aerogel *.

A liquid oxygen storage tank is filled and emptied through a pipe 5 with a valve 6 which takes a detachable metal flexible hose. Liquid oxygen enters the hose from another reservoir under
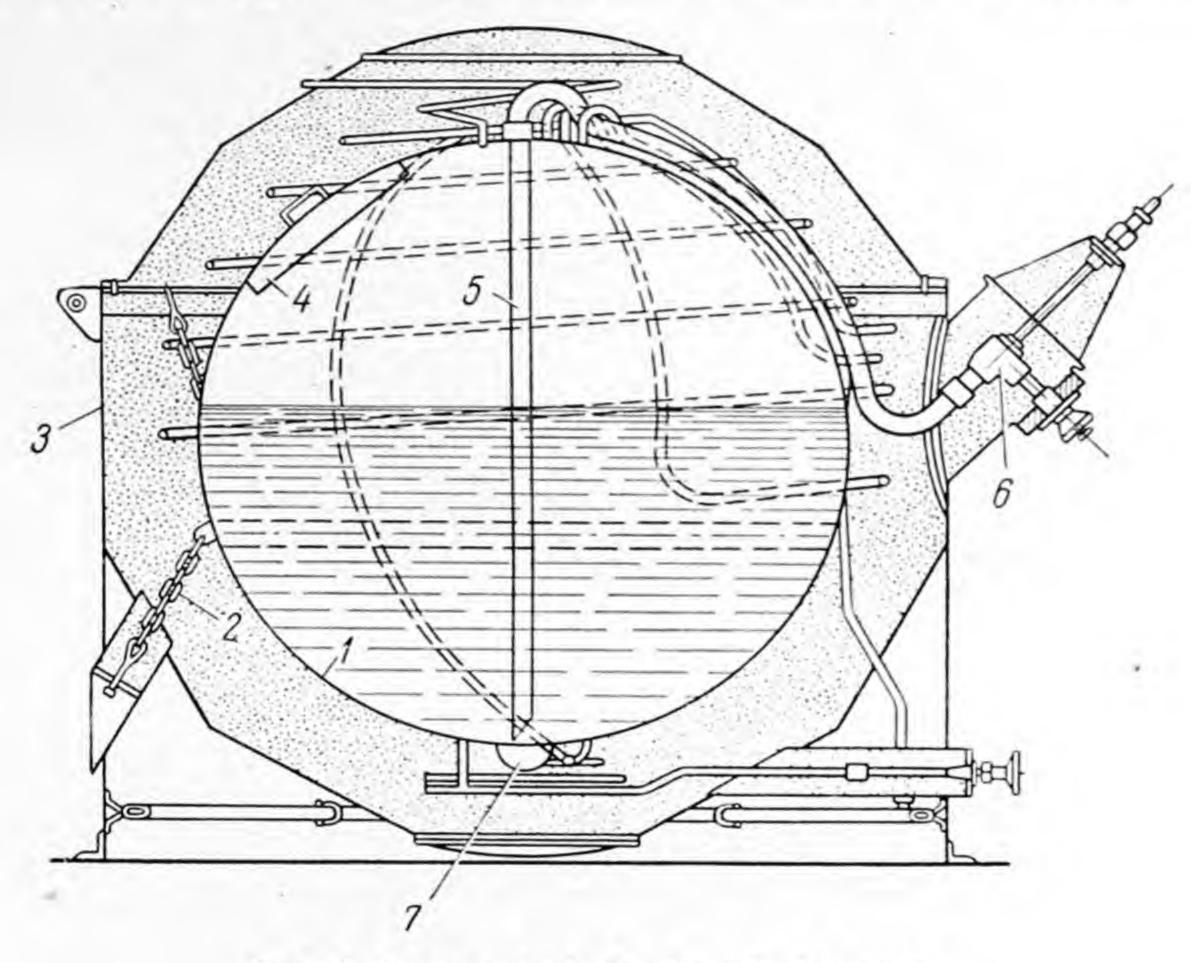
Fig. 3. Storage tank for liquefied oxygen

the action of excess pressure in the latter. Storage tanks are fitted with pressure gauges for 2 atm (gauge).

Losses of liquid oxygen in storage tanks due to evaporation range between 0.4 and 0.7 per cent per hour of the oxygen held in a tank, the percentage loss increasing as a tank is emptied.

Liquid oxygen is also stored in vacuum-insulated vessels. The

* Mypore is a light-weight highly porous heat-insulating material prepared from foamed phenol-formaldehyde resin. One cubic metre of mypore weighs 15 to 20 kg in blocks and 50 to 60 kg when crushed and compacted. Its coefficient of thermal conductivity is 0.002 to 0.025 kcal per metre per hour per degree C diff.

Aerogel is the gel of silicic acid (SiO_2), a finely divided, highly porous powder with a bulk weight of 90 to 110 kg/m³ and a thermal conductivity of 0.0118 to 0.0126 kcal per metre per hr per degree C diff.

air from the space between the walls in such a vessel is evacuated to a residual pressure of 10^{-5} to 10^{-6} mm Hg. Another variety includes vessels with what may be called vacuum-powder insulation: powdered magnesia, aerogel, or silicagel fills the space between the walls evacuated to a residual pressure of 10^{-1} to 10^{-2} mm Hg.

Vacuum-insulated and vacuum-powder insulated vessels weigh less than tanks with conventional insulation, and losses of oxygen due to evaporation in them are anywhere between 0.1 and 0.15 per cent per hour of the oxygen held.

(3) Gasification of Liquid Oxygen. Liquid oxygen is converted back to gas by *gasificators,* usually installed on the sites where oxygen comes liquefied.

High-pressure (up to 165 atm gauge) or *hot* gasificators are intended to convert to gas all the liquid oxygen charged and to fill oxygen cylinders to a high pressure.

A charge of liquid oxygen in a hot gasificator of 35 litres capacity will be enough to fill three oxygen cylinders of 40 litres capacity each to a pressure of 150 to 165 atm (gauge).

This type of gasificator can fill six cylinders an hour (36 cu m of gaseous oxygen per hour). Losses of liquid oxygen to evaporation range from 5 to 10 per cent of the charge.

Medium-pressure (up to 15 atm gauge) or *cold* gasificators store liquid oxygen under the specified pressure and convert part of the charge to gas under the same pressure in pace with its consumption. Gaseous oxygen is delivered to torches over pipelines.

A cold gasificator consists of a steel sphere inside which is placed a thin-walled brass gasometer to receive liquid oxygen.

The liquid oxygen from the gasometer is passed through a heater which consists of coils immersed in a bath filled with hot water. The water is heated by steam or electric heaters. Operation of a gasificator is controlled by means of suitable valves on a distribution board.

Instead of gasificators, liquid oxygen can be converted to gas by plunger pumps intended for pressures up to 20 atm (gauge) (the medium-pressure class) and 200 atm (gauge) (the high-pressure class). A plunger pump can discharge oxygen either into a pipeline or into oxygen cylinders.

Plunger-pump gasificators have an output of 65 to 85 litres of liquid oxygen per hour. This type of plant can fill ten oxygen cylinders of 40 litres capacity each to a pressure of up to 165 atm (gauge) every hour. Losses to evaporation do not exceed 1 per cent of the oxygen held in the storage tank of the plant.

5. Layout of an Oxygen Generating or Gasifying Plant

Though not combustible or explosive, oxygen supports combustion and explosion. For this reason any plant producing gaseous oxygen or gasifying the liquid oxygen coming from outside suppliers must comply with appropriate regulations.

An oxygen-generating plant usually consists of (a) an oxygen shop housing all the oxygen-generating equipment, auxiliary and service rooms and (b) a cylinder storage room complete with filling and storage facilities, and a cylinder testing and repair shop.

A plant with a capacity of not more than 60 cu m per hour may have both shops housed in a common, but separately standing building, or they may be placed in a common building with other production departments. In the latter case the cylinder storage room should have a floor space sufficient to accommodate not more than a day-and-a-half's output of the plant. In the case of plants with a capacity of up to 30 cu m per hour, the storage room may hold as many as 180 filled cylinders. The oxygen-generating equipment, the cylinder storage room, the filling and storage departments should be enclosed in a single-storeyed compartment leaning to an outside wall of the main building. The walls of the lean-to must be continuous and laid of a nonflammable material.

A plant with a capacity of more than 60 cu m per hour should have the oxygen-generating shop housed separately from the filling department and the cylinder storage room.

The spacing between the cylinder storage room and the oxygen-generating or gasifying shop should be not less than 20 metres for a storage room holding up to 500 filled cylinders, 25 metres for 501 to 1,500 filled cylinders, and 30 metres for over 1,500 filled cylinders.

The spacing should be not less than 50 metres between the cylinder storage room and residential quarters and not less than 100 metres between the cylinder storage room and the services.

The spacing between an oxygen-generating shop and an acetylene shop should be not less than 300 metres, with the oxygen plant located on the windward side with respect to a source of acetylene. The acetylene content where the compressor of the oxygen plant takes in air should not exceed 0.25 cu cm per cubic metre of air.

The electrical equipment of oxygen-generating plants should meet the requirements specified by appropriate electrical codes for normal locations.

All the rooms of an oxygen-generating plant may have only central heating — by hot water, steam or air.

In a storage room, the cylinders should be placed at least 100 mm away from the heaters which should be enclosed with suitable screens.

In a storage room, the filling department, the gas holder room and the filled-cylinder room should be separated from other departments by continuous walls of a nonflammable material. The service rooms and repair facilities should be located on the side of the empty-cylinder room. The filled-cylinder room should be divided by nonflammable walls into compartments holding 1,000 cylinders each. The walls may have openings for cylinder handling. Cylinders should be stored in the upright position in racks holding twenty cylinders each. The capacity of a filled-cylinder room should not exceed 3,000 cylinders. For ease of cylinder handling, the floor of the storage room should be raised 0.8 to 1 metre above the ground.

Both cold and hot gasificators may be located in separate buildings, leans-to or annexes (of at least Class II fire resistance) to the main production shop. Leans-to and annexes must be separated from the main building by continuous walls and have a separate exit.

The control panels of cold and hot gasificators must be set up in a separate room. Gas holders should be set up outside the building, behind a heavy wall.
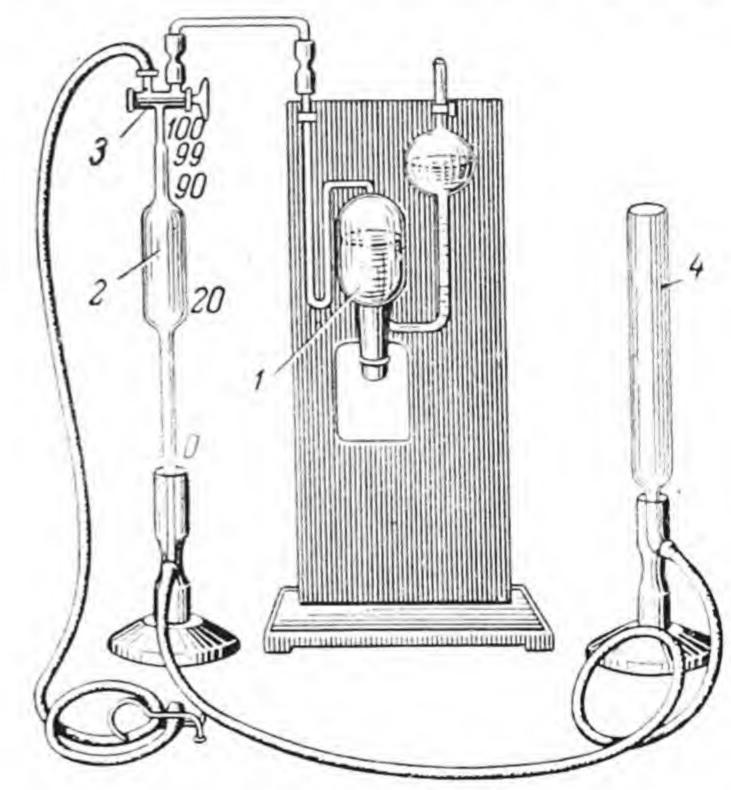
6. Oxygen Quality Control

Oxygen concentration is determined by means of an apparatus in which oxygen is absorbed by an ammonia solution of copper chloride. The apparatus (Fig. 4) consists of an absorbing pipette *1*, a measuring burette *2*, a three-way cock *3* and an equalising vessel *4*. The pipette is filled with an absorbent which is obtained by dissolving 750 grams of ammonium chloride (NH_4Cl) in one litre of water and by adding one litre of a 18-per cent aqueous solution of ammonium hydroxide (NH_4OH). Immersed in the pipette is also red copper wire 0.8 mm in diameter wound in coils 5 mm in diameter and 10 mm long. The burette has a capacity of 100 ml and is graduated in increments of 1 ml. Between the "95" and "100" graduations the burette is graduated every 0.1 ml.

For an analysis, the equalising vessel is lifted and the burette is filled with water to the "100" graduation. The equalising vessel is lowered, and the burette is filled with the oxygen under test through the three-way cock, while the water level in it is brought down to the "0" graduation. The three-way cock is turned so that

the burette communicates with the pipette; the equalising vessel is lifted again to convey the oxygen into the pipette which should be shaken for a few minutes to speed up the absorption of the oxygen by the reagent. The three-way cock is then turned so that the remaining gas can be brought back to the burette; the water level in the burette is equalised with that of the vessel, and a reading is taken from the graduations on the burette. The reading will indicate oxygen purity in per cent. The gas remaining unabsorbed in the pipette will consist of nitrogen and argon. For a

Fig. 4. Oxygen-purity test apparatus

check, the remaining gas is again passed to the pipette which is shaken several times again, and another reading is taken. If the difference between the two readings does not exceed 0.1 per cent, the analysis has been taken correctly.

Hydrogen content in oxygen obtained by electrolysis should not exceed 0.7 per cent. This is found by means of an apparatus in which the hydrogen is burned in oxygen, and the resultant reduction in the volume of the oxygen sample is determined.

According to a relevant U.S.S.R. State Standard, the moisture content of oxygen should not exceed 0.07 g/m³, which corresponds to a dew point of minus 49° C, as determined by means of a hygrometer.

CHAPTER II

CALCIUM CARBIDE AND ACETYLENE

Commercial acetylene for the gas welding industry is produced mainly by allowing water to react upon calcium carbide. For this reason in our discussion calcium carbide will precede acetylene.

The gas welding industry uses large amounts of calcium carbide for this purpose, although new processes are coming into use for acetylene generation from oil and natural gases. These new processes are based on the pyrolysis of hydrocarbon mixtures. In one of them, the methane contained in a hydrocarbon mixture is made to decompose under high heat (thermal decomposition) into acetylene and hydrogen according to the reaction $2CH_4 \rightarrow C_2H_2 + 3H_2$. The acetylene is separated from the resultant gas mixture and is further chemically treated.

1. Properties of Calcium Carbide

(1) Physical Properties. Calcium carbide CaC_2 is a chemical compound of calcium and carbon. It is a hard, opaque and crystalline substance presenting a dark-grey or brown fracture. Calcium carbide is produced by fusing together coke and lime. The reacton is endothermic and follows as described below:

$$CaO + 3C = CaC_2 + CO - 108 \text{ kcal/g-mole}; \qquad (II.1)$$

$$56.08 + 36.03 = 64.1 + 28.01$$

Thus, it takes $56.08 : 64.1 = 0.875$ kg of CaO and $36.03 : 64.1 = 0.562$ kg of carbon to produce one kilogram of chemically pure CaC_2. Theoretically, $108 \times 1,000 : 64.1 = 1,685$ kcal/kg of heat is required for this purpose.

Commercial calcium carbide contains 65 to 80 per cent CaC_2, 25 to 10 per cent CaO and up to 6 per cent other impurities (carbon, silicic acid, etc.).

The specific gravity of chemically pure CaC_2 is 2.22 at 18° C. The specific gravity of commercial calcium carbide depends on its purity and ranges between 2.8 and 2.22, decreasing with increasing purity.

Calcium carbide eagerly reacts with water, forming acetylene and slaked lime. The affinity of calcium carbide for water is so strong that it can be decomposed by water vapours in the air or crystallisation water in salts. Water reacts with calcium carbide according to the exothermic reaction:

$$CaC_2 + 2H_2O = C_2H_2 + Ca(OH)_2 + 30.4 \text{ kcal/g-mole;} \quad (II.2)$$
$$64.1 + 36.032 = 26.036 + 74.096$$

In other words, it takes $36.032 : 64.1 = 0.562$ kg of water to decompose one kilogram of chemically pure CaC_2. This reaction produces $26.036 : 64.1 = 0.406$ kg of acetylene (C_2H_2) and $74.096 : 64.1 = 1.156$ kg of slaked lime, $Ca(OH)_2$.

As measured at zero temperature and a pressure of 760 mm Hg, the specific weight of acetylene is 1.1709 kg/m³.

Theoretically, one kilogram of calcium carbide will yield $0.406 \times 10^3 : 1.1709 = 346.89$ litres of acetylene.

As measured at 20° C and 760 mm Hg, the specific weight of acetylene is 1.09 kg/m³ and the yield will be 372.5 litres per kilogram of calcium carbide. Adjusted for water vapours, the yield will actually be 380.88 litres per kilogram of calcium carbide.

When kept unsealed in the open, the ordinary moisture of the atmosphere gradually slakes calcium carbide. The rate of slaking increases directly with relative air humidity and inversely with the size of calcium carbide lumps.

In decomposing, one kilogram of chemically pure CaC_2 liberates $30.4 \times 1,000 : 64.1 = 475$ kcal of heat.

The principal impurity in commercial calcium carbide is CaO which reacts with water according to the reaction:

$$CaO + H_2O = Ca(OH)_2 + 15.1 \text{ kcal/g-mole,}$$

or

$$15.1 \times 1,000 : 56.08 = 270 \text{ kcal per kilogram}$$

$$\text{of lime } CaO. \quad (II.3)$$

For this reason, in calculating the heat developed by commercial calcium carbide, the heat given out by the lime must also be taken into account.

70-per cent commercial calcium carbide produces approximately $475 \times 0.7 + 270 \times 0.24 \approx 397$ kcal of heat per kilogram.

When there is lack of water, the temperature in the reaction zone may rise appreciably. If it rises above 200° C, CaO may be

formed according to the reaction:

$$CaC_2 + Ca(OH)_2 = C_2H_2 + 2CaO. \tag{II.4}$$

The decomposition of the calcium carbide will then be sustained by the moisture of the slaked lime, and the latter will form a dense crust on the lumps of calcium carbide. As a result, the calcium carbide will be overheated and the lumps will be covered by slaked lime. Therefore, it will be a good plan to remove the slaked lime from the carbide lumps at regular intervals so that the decomposition can proceed normally, without dangerous overheating. Otherwise this may result in the exothermic decomposition of acetylene or an explosion of the acetylene-air mixture.

(2) Acetylene Yield of Calcium Carbide. The quality of calcium carbide is mainly assessed in terms of acetylene yield per kilogram of carbide, which in turn is a function of carbide lump size. A relevant U.S.S.R. Standard specifies the two quantities as follows (Table 2).

Table 2

Acetylene Yield of Calcium Carbide

Lump size, mm	Size designation	Acetylene yield, litres/kg at 20° C and 760 mm Hg, minimum	
		Grade 1	Grade 2
2-8	2/8	255	235
8-15	8/15	265	245
15-25	15/25	275	255
25-80	25/80	285	265
Mixed	—	275	265

Calcium carbide may be supplied in lumps smaller than 25 mm in size only with the user's consent, provided he has acetylene generators which can safely use the smaller sizes of carbide lumps.

Calcium carbide of Size 8/15 and greater may have as much as 2 per cent of dust and fine particles measuring 2 mm across. The proportion of dust and smalls in Size 2/8 carbide should not exceed 5 per cent.

Calcium-carbide manufacturers using low-power furnaces (up to 1,000 kW) or low-grade raw materials are allowed to turn out calcium carbide with an acetylene yield of not below 230 litres per kilogram, irrespective of lump size.

The reduced yield of acetylene from the smaller lump sizes is explained by the fact that the smaller lumps contain more impuri-

ties (CaO, etc.) which, being less strong mechanically, are lost as wastes in crushing. In addition, the smaller lumps have a larger surface area and are, for this reason, more slaked by the moisture ordinarily present in the atmosphere during crushing and packing.

The amount of chemically pure CaC_2 in a commercial grade can be estimated by comparing the actual and theoretical yields of acetylene.

Example: The carbide grade tested gave out 255 litres of acetylene per kilogram. The theoretical yield is 0.406 kg, or 372.5 litres at 20° C and 760 mm Hg. The content of chemically pure CaC_2 will then be $255 : 372.5 \times 100 = 68.5$ per cent.

(3) The Rate of Decomposition. The rate of decomposition of calcium carbide (or the rate of gas formation) is the volume of acetylene produced by one kilogram of calcium carbide per minute (litres/kg-min). It varies with carbide grade and lump size. The smaller lumps have a higher rate due to the larger active surface area. The rate of decomposition is also affected by the temperature of the water in the generator. Fig. 5 presents experimental data showing the rate of decomposition and acetylene yield as a function of lump size and time. The maximum rate is attained and the bulk of gas is evolved during the first two to four minutes after the water has wettened the calcium carbide. At the later stage the rate of decomposition is gradually slowed down.

The decomposition of calcium carbide may also be described in terms of a reciprocal quantity — the time necessary for one kilogram of calcium carbide to be fully decomposed by water. This quantity varies directly as lump size and inversely as water temperature. Actually, account is only taken of the time necessary for 98 per cent of the carbide charge to decompose, as the rate of decomposition is then sharply reduced, and the time of decomposition of the remaining two per cent becomes meaningless for the evaluation of the reaction as a whole.

The presence of $Ca(OH)_2$ in the water slows down the decomposition of calcium carbide. Experiments have shown that when the generator water contains as much as 20 per cent $Ca(OH)_2$ at 17° C the lumps of calcium carbide are so heavily slaked on the surface that practically no gas is evolved any longer. If water temperature is raised to anywhere between 40° and 80° C, the reaction is somewhat speeded up, and only the larger lumps measuring from 25 to 80 mm across will be slaked on the surface.

For this reason the calcium carbide in the reaction zone of an acetylene generator must be stirred and the water contaminated with slaked lime changed at regular intervals.

The rate of decomposition of calcium carbide, expressed in lit/kg-min, is proportional to the amount of decomposed calcium carbide. Therefore, the former may be plotted as a function of the latter and of the time of decomposition.
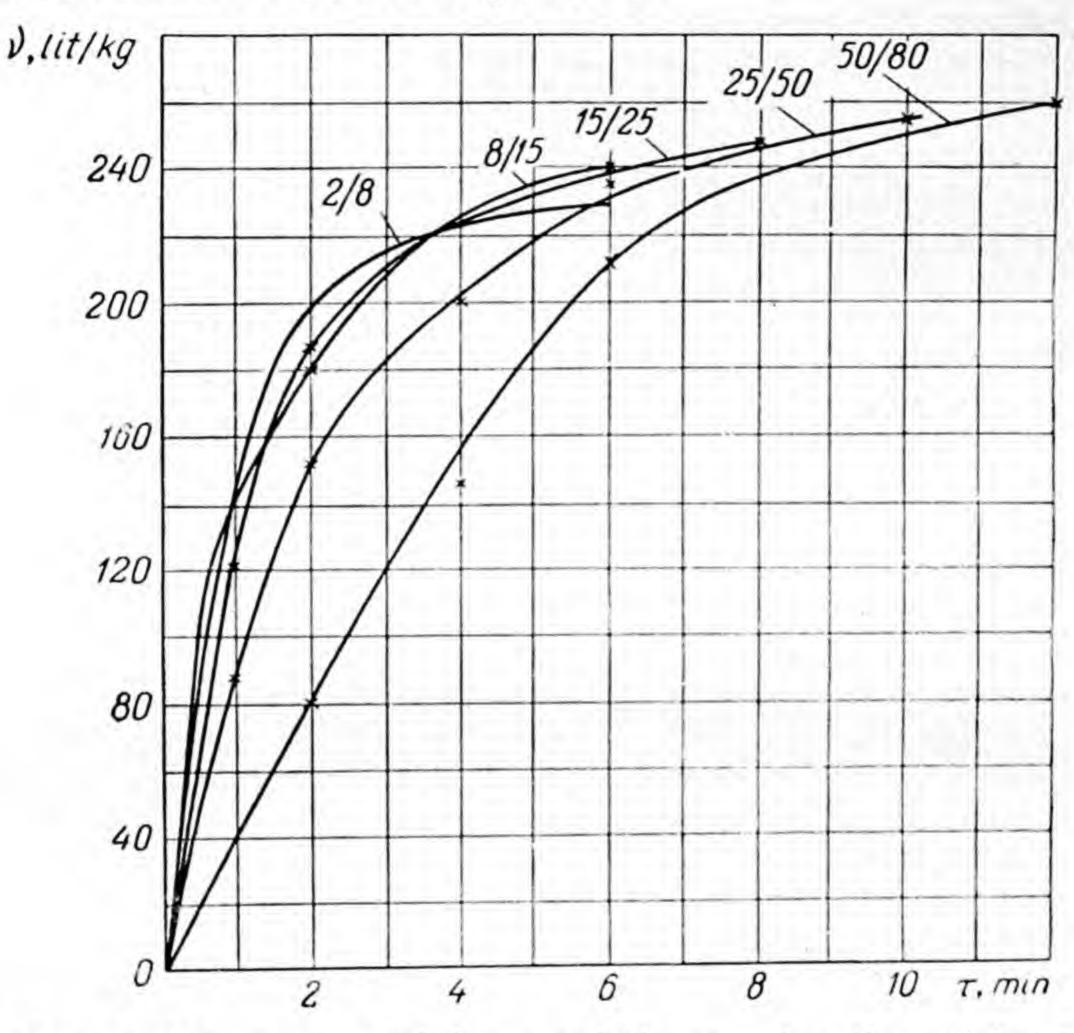
Fig. 5. Acetylene yield from carbide as a function of lump size and reaction time (at $+17°$ C and a $1:10$ carbide-to-water ratio)

Let G_o be the relative amount of calcium carbide left undecomposed, in kg/kg;

t, the time of complete decomposition of calcium carbide, in min;

τ, the time elapsed since the beginning of decomposition, also in min.

If the time τ is expressed in fractions of t, the three quantities (G_o, τ and t) may be related as follows:

$$G_o = \left(1 - \frac{\tau}{t}\right)^n. \tag{II.5}$$

Graphically, this relationship will be a parabola with an exponent n, plotted in co-ordinates $1 - G_o$ and $\frac{\tau}{t}$ (Fig. 6).

The exponent n describes the rate of decomposition. For a given method of carbide decomposition and the initial water temperature, the values of n and t are mainly dependent on the size of carbide lumps.

Fig. 6. Relative amount of decomposed calcium carbide, $1 - G_0$, plotted against the relative time of decomposition, $\dfrac{\tau}{t}$
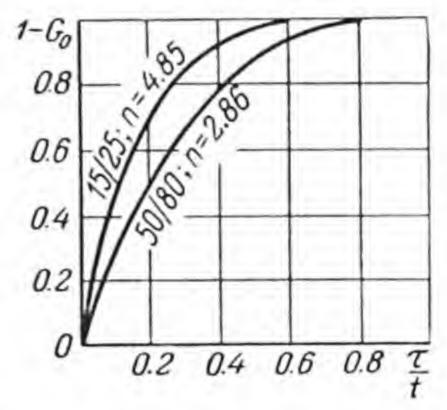
Table 3 presents the values of n and t obtained from an analysis of the empirical curves in Fig. 6 for Grade 2 carbide, a water temperature of 17° C and a water-to-carbide ratio of 10 : 1.

Table 3

Lump size, mm	Exponent, n	Time of complete decomposition, t, min	Acetylene yield, lit/kg
2-8	6.7	5.5	232.6
8-15	4.7	6.5	247.0
15-25	4.85	8.0	251.0
25-50	4.0	10.0	258.4
50-80	2.86	13.0	268.4

For Grade 1 carbide, i. e., for a greater acetylene yield, the rate of decomposition is usually higher; therefore, n is increased and t is reduced.

Let us assume that:

a = acetylene yield from calcium carbide, lit/kg;

G_c = carbide charge in a generator, kg.

Using Eq. (II.5), the amount of acetylene, V_a, in litres, evolved during the time τ, in minutes, will be:

$$V_a = G_c a (1 - G_0) = G_c a \left[1 - \left(1 - \frac{\tau}{t} \right)^n \right] \text{ litre.} \qquad (\text{II}.6)$$

The average rate of decomposition, u_{av}, will be:

$$u_{av} = \frac{a}{t} \text{ lit/kg·min (at } G_c = 1 \text{ kg).} \qquad (\text{II}.7)$$

The instantaneous rate of carbide decomposition will be:

$$u = \frac{dV_a}{d\tau} = \frac{a}{t}\, n \left(1 - \frac{\tau}{t}\right)^{n-1}. \tag{II.8}$$

The maximum rate of decomposition, u_{Max}, at $\tau = 0$, or at the beginning of the process, will be:

$$u_{\text{max}} = \frac{a}{t}\, n = u_{av} n. \tag{II.9}$$

The above relationships are used in acetylene generator design in order to determine the capacity of the gasometer.

2. Brief Outline of Calcium Carbide Production

Calcium carbide CaC_2 is produced by fusing lime CaO and carbon (coke) at 1900 to 2300° C in single- or three-phase electric arc furnaces which may range from 50 to 30,000 kW in power rating. Theoretically, it takes 1,965 kWh of electricity to produce one ton of chemically pure calcium carbide. Actual energy consumption is greater owing to heat losses in the furnace and varies with furnace capacity. For large furnaces of 7,500 to 30,000 kW, energy consumption is 3,200 to 2,800 kWh per ton of calcium carbide; for medium furnaces of 1,000 to 7,500 kW the figure is 4,000 to 3,200 kWh per ton of carbide; and for small furnaces rated at up to 1,000 kW energy consumption rises to as high as 4,000 to 7,000 kWh per ton of carbide.

The principal raw material in calcium carbide production is limestone ($CaCO_3$). It is first roasted in lime kilns to produce quick lime (CaO). The quick lime thus produced is mixed with ground coke and anthracite in right proportion to make up the charge. The charge is fed into the hoppers of a carbide furnace whence it is automatically loaded into the furnace shaft. In the furnace the charge is melted to form molten CaC_2. At regular intervals the molten carbide is tapped from the furnace into cast-iron moulds or into a continuously rotating water-cooled crystalliser in which the carbide solidifies into ingots. Carbide ingots are then taken to a special department where they are cooled and crushed. The crushed carbide is next fed to the sizing department where it is classed by lump size, discharged into hoppers, and is then packed in roof-iron drums in which it is shipped to users.

The lime and coke used for carbide production must be of high purity, or their impurities will be carried over into the finished carbide, thus impairing its quality. In addition, raw materials of poor quality entail greater energy consumption.

The coke and anthracite that go to make calcium carbide should contain not more than 0.04 per cent phosphorus and not over 1.5 per cent sulphur. These impurities are most harmful, as they are fully carried into the carbide as Ca_3P_2 and CaS, respectively, whence they find their way into the acetylene, forming PH_3 and H_2S, both harmful impurities.

Production of one ton of calcium carbide requires, on the average, 950 to 1,000 kg of lime, 600 to 610 kg of coke and anthracite, and 40 to 70 kg of electrode material.

3. Control of Carbide Quality

The quality of the calcium carbide shipped to the user is checked by the manufacturer who must guarantee that his product meets the respective specifications. The user, however, may do the checking for himself, following the procedure established by a relevant standard.

Lump size is determined on the basis of a screen analysis of 1-kg sample.

Acetylene yield is ascertained on the basis of 1-kg samples taken from each drum. A sample is partially decomposed in a suitable apparatus and the volume of the evolved acetylene is measured.

This volume can be found by the equation:

$$V = \frac{V_1 (p_0 - p_w) (273 + 20)}{760 (273 + T) G},$$ (II.10)

where V = acetylene volume, lit/kg CaC_2;

V_1 = acetylene volume in the measuring vessel, litres;

G = weight of a sample, kg;

p_w = water vapour pressure at the temperature of the test, mm Hg;

T = acetylene temperature during the test, deg. C;

p_0 = barometric pressure, mm Hg.

This pressure can be reduced to normal conditions by subtracting from the barometric pressure 1 mm Hg when $T = 6$ to $12°$ C, 2 mm Hg when $T = 13$ to $20°$ C, 3 mm Hg when $T = 21$ to $28°$ C and 4 mm Hg when $T = 29$ to $35°$ C.

Three measurements are taken and their arithmetical mean is found, provided the difference between them does not exceed 10 litres.

The content of PH_3 and H_2S is determined by a chemical laboratory, using the procedure outlined in a relevant standard and the same samples which are used for acetylene yield determination.

3 –413

The maximum content of PH_3 and H_2S for acetylene produced from commercial carbide is 0.08 per cent and 0.15 per cent (by volume), respectively.

4. Storage of Calcium Carbide

Unless it is kept and shipped in hermetically sealed drums of roof iron at least 0.51 mm thick, calcium carbide will be slaked by the moisture of the atmosphere and evolve acetylene which may make an explosive mixture with air. In practice, two sizes of drums are used:

Carbide weight, kg	Drum diameter mm	height mm
100	420	665
130	435	700

Instead of expendable roof-iron drums, calcium carbide may be shipped in returnable 60-litre and 110-litre drums made from low-carbon sheet steel and fitted with hermetically sealed lids with rubber gaskets. In the U.S.S.R. these drums are covered by a special standard, "Drums for Chemical Products".

Drums holding calcium carbide must be stored in dry, well ventilated locations. Carbide storage rooms must be located where they are safe from being flooded by ground water. Their spacing is given below:

	Least spacing, metres from production buildings	from residential quarters
Small storages (up to 2 tons) . . .	10	15
Medium storages (2 to 20 tons) . .	15	25
Large storages (over 20 tons) . . .	20	40

Storage buildings must meet Class 1 or Class 2 fire-safety requirements, and have neither water nor heating pipelines inside them. Should a fire break out in a carbide storage room, it must be put out with CO_2 fire extinguishers and sand, and not with water. Carbide drums may be opened with tools which will not produce sparks (a brass chisel and hammer) or with special knives resembling tin-openers. Every time some carbide is taken from an opened drum, it should be closed again with a lid and rubber gasket. If this cannot be done, the remaining carbide must be emptied into hermetically closing cans. When handling calcium carbide, personnel in charge should wear tarpaulin gauntlets or rubber gloves.

5. Properties of Acetylene

(1) **Physical and Chemical Properties.** Acetylene is a hydrocarbon of the $C_n H_{2n-2}$ unsaturated group. It has the chemical formula $C_2 H_2$ and the structural formula $H-C\equiv C-H$.

Under normal conditions acetylene is a colourless gas. Commercial acetylene has a pungent, nauseating odour which it owes to the impurities present in it, such as phosphine (PH_3) and hydrogen sulphide ($H_2 S$). Acetylene has a density of 1.091 kg/m^3 at 20° C and 760 mm Hg, and of 1.171 kg/m^3 at zero and 760 mm Hg.

At 20° C and 760 mm Hg, its heat capacity at constant pressure, c_p, is 0.402 kcal/kg-deg; and at constant volume, c_v, 0.325 kcal/kg-deg. The ratio of the former to the latter is 1.235. Its gas constant is $R = 32.58$ кg-m/kg-deg. Its specific gravity as compared with air is 0.9056.

Its critical pressure, p_{cr}, is 61.6 atm (abs), and its critical temperature, T_{cr}, is +35.9° C. Under atmospheric pressure, acetylene can be liquefied at a temperature of −82.4° C to −83.6° C.

Acetylene solidifies when cooled down to −85° C or below. The density of solid acetylene is 0.76 kg/cm^3. Under certain conditions liquid and solid acetylene can explode due to friction, impact or detonation.

Complete combustion of acetylene follows the reaction:

$$C_2 H_2 + 2.5 O_2 = 2CO_2 + H_2 O + H. \tag{II.11}$$

In other words, for complete combustion one volume-unit of acetylene requires two-and-a-half volume-units of oxygen.

According to some investigators, acetylene has the highest calorific value of 14,000 $kcal/m^3$ at zero temperature and 760 mm Hg.

The low heat value, at zero temperature and 760 mm Hg, may be taken equal to 13,500 $kcal/m^3$.

The heat of combustion of acetylene is the sum of the heat of its exothermic decomposition and the heats of combustion of the original carbon and hydrogen.

As acetylene decomposes according to the reaction:

$$C_2 H_2 \rightleftarrows 2C + H_2 + H_0 \tag{II.12}$$

the heat of decomposition is $H_0 = 53.9$ kcal/g-mole or $53.9 \times 1,000 : 26.036 = 2,070$ kcal/kg.

The heat of combustion of C and H_2 is:

$$C + 0.5 O_2 \rightleftarrows CO + 26.84 \text{ kcal/g-mole};$$
$$C + O_2 \rightleftarrows CO_2 + 94.45 \text{ kcal/g-mole};$$
$$CO + 0.5 O_2 \rightleftarrows CO_2 + 67.61 \text{ kcal/g-mole};$$
$$H_2 + 0.5 O_2 \rightleftarrows H_2 O \text{ (vapour)} + 57.84 \text{ kcal/g-mole};$$
$$H_2 + 0.5 O_2 \rightleftarrows H_2 O \text{ (liquid)} + 68.35 \text{ kcal/g-mole};$$

3*

Hence:

$H_{high} = 53.9 + 2 \times 94.45 + 68.35 = 311.15$ kcal/g-mole,

or $311.15 \times 1,000 : 26.036 = 11,950$ kcal/kg,

or $11,950 \times 1.171 =$ approx. 14,000 kcal/m³.

This agrees well with the empirical heat value quoted for acetylene above.

A similar calculation for the low heat value of acetylene at 0° and 760 mm Hg (assuming that the H_2O is vapour) will give

$$H_{low} = \frac{300.64 \times 1,000 \times 1.171}{26.036} \quad 13,500 \text{ kcal/m}^3$$

Acetylene is a highly endothermic compound, i. e., it absorbs heat during its production and liberates heat when decomposed. Its explosibility depends on the conditions under which acetylene is used.
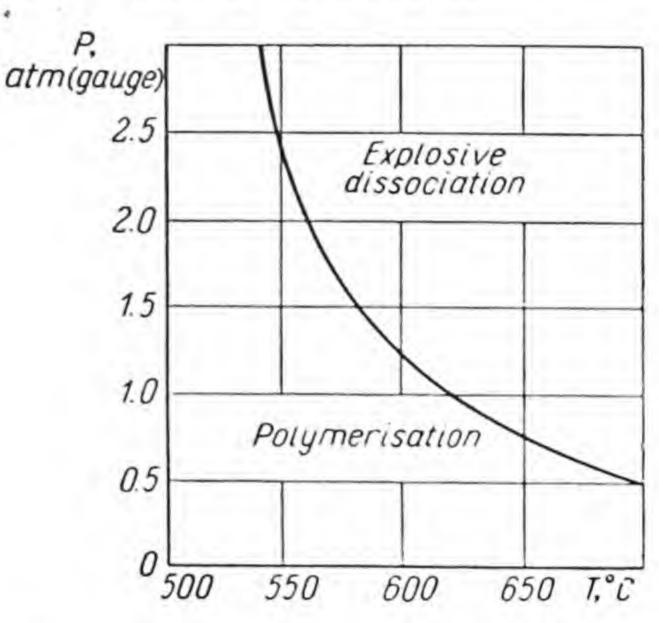
(a) Explosibility of Pure Acetylene. For pure acetylene, its explosibility is above all a function of both its pressure and temperature. It may also be strongly affected by the presence of impurities, moisture and catalysts, by the nature of the source of ignition, the size and shape of its container, conditions of heat transfer, etc.

The multitude of factors which may bring about an explosion of acetylene can also be explained by the fact that the explosion may be preceded by what is known as *polymerisation of acetylene*. The polymerisation consists in that at a temperature above 200° or 300° C the acetylene molecules are regrouped into more complex compounds, such as C_6H_6 (benzene), C_8H_8 (styrene), $C_{10}H_8$ (naphthalene), C_7H_8 (toluene), etc. The polymerisation is always exothermic. For example, $3C_2H_2 \rightarrow C_6H_6 + 150.58$ kcal/g-mole. The character and rate of polymerisation have a decisive effect on the dissociation of the acetylene. In turn, the rate of polymerisation increases directly with the temperature of the gas. Therefore, the evolved heat promotes further polymerisation, which may, in the long run, result in an explosion. If much heat is abstracted in the process, no explosion will result, and the reaction will be limited to that of polymerisation.

Fig. 7 shows arbitrarily delineated areas of polymerisation and explosive dissociation for acetylene. As can be seen from the chart, at 540° C and below and a pressure of less than 3 atm (gauge) the process is predominantly that of polymerisation. At 1.5 atm (gauge) and over 580° C explosive dissociation may take place. As the pressure increases, a progressively lower temperature is required for the polymerisation to develop into explosive dissociation. In the light of this consideration, existing regulations limit

the pressure of gaseous acetylene in acetylene generators to 1.5 atm (gauge). At the same time, it is assumed that a temperature of 580 to 600° C at which explosive dissociation is likely to take place under the specified pressure can hardly be attained in the generating chamber or the gas holder. An explosion of acetylene raises the pressure in the vessel containing the products of acetylene decomposition by a factor of eleven or twelve.

Fig. 7. Areas of polymerisation and explosive dissociation of acetylene

The presence of water vapours in acetylene appreciably checks the tendency of acetylene towards explosive dissociation. A mixture of 1.15 part (by volume) of acetylene and 1 part (by volume) of water vapour will not explode. On the other hand, some substances, such as copper oxide, are catalysts and, as such, may shift the reaction towards explosive dissociation.

Below are the lowest temperatures and materials in contact with which acetylene exploded in an experiment under a pressure of 4 atm (gauge):

Iron filings	520° C
Brass filings	500-520° C
Calcium carbide	500° C
Aluminium oxide . . .	490° C
Copper filings	460° C
Activated coal	400° C
Rust (iron hydroxide) .	280-300° C
Iron oxide	280° C
Copper oxide	240° C

Given sufficient pressure, temperature and time, acetylene can react with heavy metals, such as copper and silver, forming so-called acetylides which are explosive substances. For this reason

acetylene pipelines and fittings must not be fabricated from alloys containing more than 70 per cent copper. Appreciable quantities of copper acetylide (Cu_2C_2 or $12C_2Cu \cdot H_2O$) can only be formed when damp unpurified acetylene is allowed to be in prolonged contact with pure copper.

(b) Explosibility of Acetylene in Mixtures with Other Gases. An oxygen-carrying gas added to acetylene enhances its explosibility. Mixtures of acetylene and air or pure oxygen may explode even under atmospheric pressure, should the temperature anywhere in the gas volume reach that of self-ignition. For an air-acetylene mixture, self-ignition lies between 305 and 470° C, and for an oxy-acetylene mixture, between 297 and 306° C. The explosibility of such mixtures is mainly dependent on their acetylene content. Table 4 gives the limits or explosibility for acetylene and other fuel gases in a mixture with air or oxygen.

Table 4

Limits of Explosibility for Mixtures of Fuel Gases and Air or Oxygen

Gas	Percentage proportions, by volume			
	gas	air	gas	oxygen
Acetylene	2.2	81.0	2.8	93.0
Hydrogen	3.3	81.5	4.65	93.9
Carbon monoxide . . .	11.4	77.5	15.5	93.9
Methane	4.8	16.7	5.0	59.2
Ethylene	2.75	28.6	4.1	61.8
Ethane	3.12	15.0	4.1	50.5
Propane	2.17	9.5	—	
Butane	1.55	8.4	—	
City gas	3.8	24.8	10.0	73.6
Coke-oven gas	7.0	21.0	—	
Natural gas	4.8	14.0	—	
Oil gas	3.5	16.3	—	
Petrol vapour	0.7	6.0	2.1	28.4
Kerosene vapour	1.4	5.5	—	

Increased pressure reduces the point of self-ignition and mixtures become more explosive. For example, a mixture containing 50 per cent air, when compressed to 2.5 atm (gauge), will ignite at 300° C.

Air-acetylene mixtures containing 7 to 13 per cent acetylene and oxy-acetylene mixtures containing about 30 per cent acetylene are most explosive. In a 1-to-1 oxy-acetylene mixture the

shock wave propagates at 3,000 m/sec. When detonated, such mixtures develop pressures in excess of 350 atm (gauge) and, sometimes, as high as 600 atm (gauge).

When added to acetylene, gases which do not react with it (such as nitrogen, carbon monoxide, methane, hydrogen, oil gas, etc.) reduce its explosibility. A similar effect is obtained when acetylene is dissolved in liquids. This can be explained by the fact that the acetylene molecules are separated by those of the diluting gas or of the solvent, and the propagation of an explosion is hampered. Mixtures containing 45 per cent acetylene and 55 per cent methane, or 18 per cent acetylene and 82 per cent hydrogen will become explosive only under a pressure of 20 atm (gauge) or more. An addition of carbon dioxide also reduces the explosibility of acetylene. Acetylene dissolved in acetone will explode only under pressures upwards of 10 atm (gauge). Addition of these gases to acetylene and its dissolution in water or acetone can reduce its explosibility (to phlegmatise it) to a point where it can be safely produced, stored and used under a pressure of 15 to 25 atm (gauge).

(2) Impurities in Acetylene. (a) *Air* finds its way into acetylene when a generator is being charged with calcium carbide. In addition, some air is always dissolved in the feed water of the generator and adsorbed on the surface of the carbide lumps. Air is a harmful impurity as it aggravates the explosibility of acetylene. Every measure should therefore be taken in order to obtain acetylene with as low air content as practical. Ordinarily, stationary acetylene generators produce acetylene containing under 0.5 per cent air, and portable generators 1 to 1.5 per cent. The acetylene withdrawn immediately after a generator has been charged with calcium carbide may have as much as 45 per cent air or even more. Acetylene containing more than 10 per cent air should be discharged into the atmosphere through the escape or relief devices of the generator, rather than burned in torches.

(b) *Water vapour* is always present in the acetylene supplied to a torch from a generator. The moisture content of acetylene under conditions of complete saturation depends on the temperature of the gas:

°C	5	10	15	20	30	40	50	60	70	80
Moisture content, g/m³, at 760 mm Hg	6.8	9.4	12.8	17.2	30.1	50.8	82.3	129.3	196.6	290.7

Water vapour is an undesirable impurity as it reduces flame temperature to a certain extent because some heat is spent to partly dissociate the vapour in the welding zone. Experiments

have shown that the welding rate falls by as much as 3 to 10 per cent when the moisture content in the acetylene is allowed to rise to 25 to 70 g/m³. The acetylene taken from a cylinder contains no moisture, as the latter remains entrapped in the acetone and the porous filler in the cylinder.

(c) *Acetone vapour* (see Sec. 6). When the gas is discharged from an acetylene cylinder, some acetone is withdrawn with the gas in the form of acetone vapour. This proportion increases with the temperature and with the rate of gas flow from a cylinder. Usually, it is 45 to 50 g/m³, rising to 100 g/m³ or more in the most unfavourable cases. There is no information, however, that acetone vapour has a detrimental effect on the weld or welding operation. The only reason why it is worth while keeping the proportion of acetone vapour in the gas to a minimum is economy of acetone which otherwise has to be replenished when refilling acetylene cylinders.

(d) *Phosphine* (PH_3) is formed during acetylene generation by the reaction of the water on the calcium phosphide contained in commercial carbide. Two reactions may take place:

$$P_2Ca_3 + 6H_2O = 2PH_3 + 3Ca(OH)_2 \qquad (II.13)$$

or

$$P_2Ca_2 + 4H_2O = P_2H_4 + 2Ca(OH)_2 \qquad (II.14)$$

Raw acetylene may have as much as 0.03 to 18 per cent phosphine (by volume). The method of acetylene production does not affect the phosphine content which only depends on the purity of the original calcium carbide. Phosphine is a harmful impurity as, under certain conditions, it may cause the air-acetylene mixture to self-ignite (when, say, a generator is charged with a fresh load of calcium carbide).

Gaseous phosphine PH_3 can self-ignite in the presence of air at 100° to 200° C. "Liquid" hydrogen phosphide P_2H_4 in the presence of air can self-ignite even at below 100° C. When a generator is started, there always is some air inside it. Therefore, should the carbide charge be heated to about 200° C, the phosphine may well self-ignite, setting off an explosion of the air-acetylene mixture. For this reason the relevant standard limits the PH_3 content in acetylene to 0.08 per cent (by volume). The actual content may be as high as 0.09 per cent owing to an error of ± 0.01 per cent unavoidable in determinations.

Cases are known when the air-acetylene mixture in "water-to-carbide" generators exploded owing to a phosphine content of 0.14 per cent (by volume) in the acetylene.

Of course, a phosphine content of 0.14 per cent will not necessarily lead to an explosion, provided the situation is not aggra-

vated by heat concentration, quick temperature rise in the generating chamber, presence of sufficiently much air, inclusions of phosphorus combinations in the calcium carbide, etc.

(e) Hydrogen sulphide H_2S is also a harmful impurity in acetylene. It is mainly produced by the reaction of water on the calcium sulphide, aluminium sulphide and calcium carbosulphide present as impurities in any grade of commercial carbide.

Its content depends on the manner in which acetylene is generated. When it is produced by dropping carbide in an excess of water (the carbide-to-water process), the greater proportion of sulphur remains as H_2S in the sludge. On the other hand, in water-to-carbide and recession-type generators, where water is allowed to come in contact with an excess of carbide, heat abstraction is insufficient and the temperature in the generating chamber may rise, thus increasing the H_2S content in the acetylene. The H_2S content is usually specified within 0.08 to 1.5 per cent (by volume).

When the weld metal picks up some phosphorus or sulphur, its quality is usually impaired. Experiments made on low-carbon steel have failed, however, to prove that the same is the case with the PH_3 and H_2S present in acetylene. Indeed, the weld metal remained practically unaffected when raw acetylene carried as much as 0.1 per cent PH_3 and 0.1 (even 0.25) per cent H_2S.

6. Dissolved Acetylene

The solubility of acetylene in one litre of solvent at 1 atm (abs) and 15° C is as follows:

In water	1.15 litres
In benzene	4.0 litres
In petrol	5.7 litres
In commercial methylacetate	14.8 litres
In acetone (CH_3COCH_3)	23.0 litres

With an increase in temperature the solubility of acetylene in water and acetone is decreased (Table 5).

The solubility of acetylene in water is utilised in acetylene generators. It has also to be taken into account in assessing the actual yield of acetylene from calcium carbide.

The outstanding ability of acetone to absorb enormous quantities of acetylene in proportion to the pressure applied makes it possible to increase the actual capacity of cylinders used for acetylene storage and shipment.

In industrial practice the term "dissolved" applies to compressed acetylene pumped into steel cylinders which are filled with a porous mass soaked with acetone. The acetylene is dissolved

Table 5

Effect of Temperature on Solubility of Acetylene

(a) In Water

°C	0	5	10	15	20	30	40	50	60	70	80
Litres of acetylene dissoluble in one litre of water	1.73	1.49	1.31	1.15	1.03	0.84	0.65	0.50	0.37	0.25	0.15

(b) In Acetone

°C	−20	−15	−10	−5	0	+5	+10	+15	+20	+25	+30	+35	+40
Litres of acetylene dissoluble in one litre of acetone	52	47	42	37	33	29	26	23	20	18	16	14.5	13

under pressure in the acetone filling the pores. When the valve is opened, the pressure in the cylinder is reduced, and the acetone releases gaseous acetylene.

(1) Porous Fillers. If acetylene were compressed into an ordinary empty cylinder, it might explode before it had reached any useful pressure. To prevent this, acetylene cylinders are filled with a highly porous substance which divides the space within the cylinder into minute cells. As a result, should an explosion take place, the products of dissociation will be better cooled, some of the explosive energy will be absorbed by the narrow capillaries in the filler, and the dissociation will thus be confined to the spot where it originates.

Another purpose of the porous filler is to speed up the absorption of the acetylene by the acetone when a cylinder is being filled and to facilitate the release of the acetylene from the acetone when the gas is discharged from the cylinder. For its action the acetone-soaked porous filler depends on the much greater area of contact between the gas and the solvent. When properly chosen, the filler additionally increases the capacity of a cylinder at a given pressure and temperature owing to the adsorption of the gas on its surface.

Fillers are made from a variety of mixtures of inert, highly porous and light-weight materials, such as pumice, infusorial

earth, charcoal, activated coal, asbestos fibre, etc. In the Soviet Union wide use is made of a porous filler developed by D. L. Glizmanenko and S. Y. Miloslavsky which consists of granulated activated charcoal with a grain size of 1 to 3.5 mm. This filler, designated the БАУ is of light weight (300 to 310 g/lit) and high porosity (80 per cent).

A commercial porous filler should meet a number of requirements:

(a) the ingredients should be chemically inert with respect to acetylene, solvent and steel;

(b) the filler should be sufficiently strong to stand wear and tear in service, should not settle down or form cavities in cylinders;

(c) the porosity of the filler should be not less than 74 per cent; its capillarity should be sufficient for the uniform distribution of the acetone and take as much acetylene as possible;

(d) the filler should be light in weight (not more than 350 to 400 g/litre);

(e) the filler should reliably localise an explosion of the acetylene in a cylinder, prevent its propagation through the entire gas quantity, and be insensitive to the elevated temperatures which may be developed in a cylinder by local dissociation of the acetylene.

Whether or not a filler meets these requirements is ascertained on the basis of a series of production and explosion tests.

(2) Determination of the Gas Quantity in a Cylinder. Most accurately the gas quantity in a cylinder can be determined by weighing it before and after its use and by computing the gas volume by the equation

$$V = \frac{G_2 - G_1}{\gamma},$$ (II.15)

where V is the volume of gas in a cylinder at 20° C and 760 mm Hg in cu m;

G_1 = the weight of the cylinder before it was filled with acetylene, in kg;

G_2 = the weight of the cylinder after it was filled with acetylene, also in kg;

γ = the specific weight of acetylene at +20° C and 760 mm Hg, equal to 1.091 kg/m³.

(3) Factors Affecting the Pressure and Gas Quantity in a Cylinder. The pressure in an acetylene cylinder varies with temperature. As the temperature drops, the solubility of acetylene

increases, and the pressure in the cylinder falls. Acetylene cylinders are, therefore, filled so that at 20° C the pressure will not exceed 16 atm (gauge). Actually, the filling operation is stopped at a higher pressure of 22 to 25 atm (gauge) to make up for the increase in the temperature inside the cylinder owing to the heat given up by the acetylene dissolving in the acetone.

The following table gives an idea of how the pressure in an acetylene cylinder varies with temperature:

Ambient temperature, °C	−10	−5	0	+5	+10	+15	+20	+25	+30	+35	+40
Pressure, atm (gauge)	7	8	9	10.5	12	14	16	18	20	22.5	25

When a cylinder is heated, the acetylene-acetone solution in it expands. After the solution fills all free space in the cylinder, any further heating may raise the pressure to a point where the cylinder may fail. To avoid this, acetylene cylinders are filled with acetone so as to have a 6 to 10 per cent excess of acetylene (of the cylinder capacity). Then heating to 40° C will not raise the pressure in the cylinder above 25 atm (gauge).

Usually, an acetylene cylinder holds an average of about 0.11 to 0.12 kg (or 0.1 to 0.11 cu m) of acetylene per litre of its water capacity. The quantity of acetylene dissolved in acetone varies within broad limits with the quality of the solvent. The latter should be Grade 1 commercial acetone (CH_3COCH_3) of sp. gr. 0.795 and with a boiling point of 56° C. Every precaution must be taken to prevent any water from getting into a cylinder together with acetylene. This may happen when the latter has not been properly dried. Accumulating in a cylinder, the moisture can markedly reduce its acetylene capacity. As little as 2.7 per cent water added to acetone reduces the solubility of acetylene by as much as 13 per cent, while an addition of 10.7 per cent water cuts it down by 35 per cent.

Cylinders with too much water in them have to be given a special treatment which consists in removing the water-acetone mixture from the porous filler in an electric oven (Fig. 8). This oven is a cylindrical enclosure made from two lengths of steel pipe placed concentrically. The space between them is filled with slag wool for thermal insulation. Inside the enclosure are placed heating elements wound with Ni-Cr wire 1.4 mm in diameter. The temperature inside the oven is measured with a thermocouple and should not exceed 250° C. The acetone-water vapours leaving the

cylinder are made to pass through a pipe into a cooler where they condensate. This operation lasts for six hours, after which the cylinder is evacuated, allowed to cool and filled with pure acetone. The oven, rated at 4 kW operates on 220 V and consumes 18 kWh per cylinder.
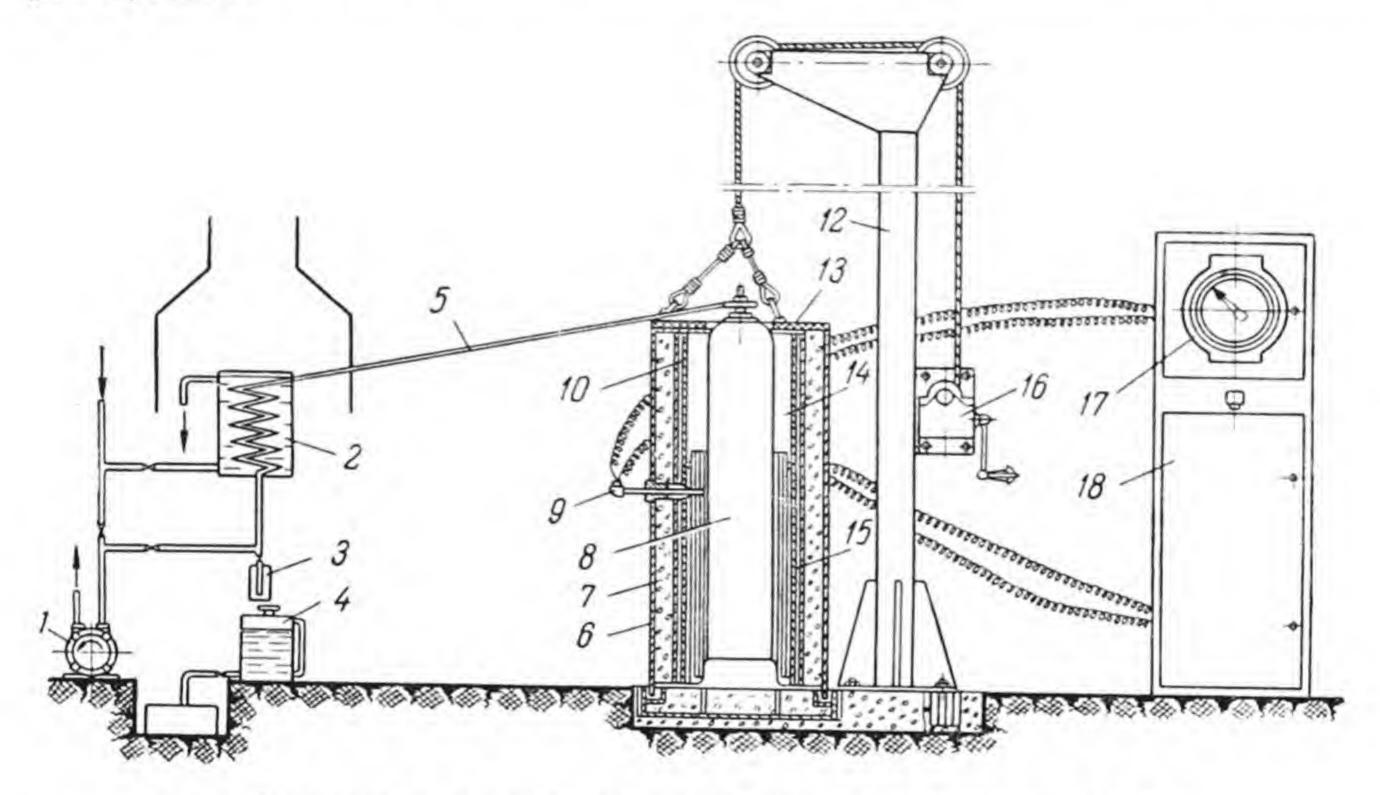
Fig. 8. A set-up for the reconditioning of acetylene cylinders:
1 — vacuum pump; *2* — cooler; *3* — hydrometer vessel; *4* — metal vessel *5* — acetone-water mixture off-take tube; *6* — outer shell; *7* — space filled with mineral wool; *8* — acetylene cylinder; *9* — thermocouple; *10* and *14* — inner casings; *12* — upright; *13* — asbestos-cement cover; *15* — electric-heating elements *16* — winch; *17* — valve potentiometer; *18* — control console

(4) Quality of Dissolved Acetylene. The quality of dissolved acetylene shipped to the user should meet the requirements of the standard, "Commercial Dissolved Acetylene". According to this standard, a sample of acetylene taken from a cylinder should contain not more than 2 per cent air and other impurities poorly soluble in water, not over 0.02 per cent phosphine (PH_3), and not exceeding 0.05 per cent H_2S (by volume). The pressure in a cylinder should correspond to the ambient temperature (see Sec. 6 (3)).

(5) Advantages of Dissolved Acetylene. Production of dissolved acetylene is only possible in special shops using suitable equipment and cylinders and involves additional expenditures on filling facilities. Nevertheless it offers the following advantages

over the free acetylene produced in portable generators right on a gas-welding site:

(a) high purity of the gas withdrawn from a cylinder; absence of moisture and insignificant content of harmful impurities;

(b) higher pressure essential for the steady operation of torches, constant composition of the combustible mixture under the most adverse conditions, such as heavy heating of the tip and long pipelines taking the gas to torches;

(c) compact size and cleanliness of the welding station owing to the absence of carbide, acetylene generator, water and sludge-discharging and storing facilities;

(d) readiness of acetylene supply at a moment's notice;

(e) safety of use in hot shops, boiler-rooms and other locations where an acetylene generator presents a hazard of explosion owing to possible leakage of free acetylene;

(f) continuous supply of acetylene at low ambient temperatures where an acetylene generator may fail owing to the water freezing in the water-seal and hoses;

(g) better utilisation of carbide, as stationary generators are more economical than portable ones;

(h) no additional labour to attend to portable acetylene generators.

(6) New Processes for Production of Dissolved Acetylene. The Autogenous Welding Institute in the Soviet Union has developed a high-pressure acetylene generator which dispenses with a compressor for cylinder filling. In this generator the calcium carbide is decomposed in a closed space and generates acetylene compressed to 30 kg/cm². For safety reasons water is made to circulate continuously through the generating chamber. The acetylene leaving the generator is purified, dried and charged into cylinders. Field operation of this type of generator has proved successful.

To make the process more economical and capable of wider use, the Autogenous Welding Research Institute has built this generator into a combination plant which can produce either concurrently or separately both high-pressure dissolved and medium-pressure free acetylene.

Y. V. Dalago and G. F. Chepelyugin have utilised the marked increase in acetylene solubility in acetone with decreasing temperature in a plant which charges the acetylene produced in a medium-pressure generator (0.4 to 0.5 kg/cm²) into cylinders cooled to 60 or 70° C below zero. Cylinders are cooled by the waste cold resulting from the evaporation of liquid oxygen in cold gasificators.

When cylinders thus filled are then raised to the ambient temperature, the pressure in them rises to the magnitude obtainable with compressors. This method can be employed where the cold is wasted and the user has medium-pressure acetylene generators.

7. Control of Acetylene Quality

Acetylene may be tested for quality either by the supplier or by the user. This is done by analysing it for air content and the presence of PH_3 and H_2S.

When acetylene is generated in stationary equipment, samples are taken from discharge pipelines. If acetylene is supplied dissolved in cylinders, every twentieth cylinder in each lot is tested for acetylene quality. Samples are taken from each cylinder thus selected.

The air content is determined by passing a stream of running water through a burette holding 100 ml of acetylene. The acetylene dissolves in the water, while the air remains in the burette as a gas owing to the fact that it has a lower solubility. The volume of the air is determined as the difference between the water levels before and after the acetylene is dissolved.

The PH_3 and H_2S *contents are determined* by techniques of quantitative analysis. PH_3 can be determined qualitatively by placing a piece of filtering paper soaked with a 5-per cent solution of silver nitrate ($AgNO_3$) in a stream of the acetylene under test; the paper will grow dark in the presence of PH_3.

H_2S can be determined qualitatively by placing a piece of filtering paper soaked with a solution of mercury chloride ($HgCl_2$) in a stream of the acetylene to be tested; in the presence of appreciable amounts of H_2S the paper will grow light in colour.

CHAPTER III

ACETYLENE GENERATORS

1. Purpose and Classification

The name "acetylene generator" is given to a device which produces acetylene from calcium carbide and water.

Acetylene generators may be classed according to (1) capacity, (2) service, (3) process control and (4) acetylene pressure in the generator or gas holder.

According to capacity, they are classed into those of 0.8, 1.25, 2, 3.2, 5, 10, 20, 40 and 80 m³/hr. Some of the flame metal-working processes require acetylene generators of 150 m³/hr capacity. Acetylene generators producing more than 150 cu m per hour are employed in the chemical industry and are designed and built to special requirements.

According to service, acetylene generators are divided into portable and stationary. The former are usually of up to 3.2 m³/hr capacity, inclusive. In some cases, such as in the gas-pressure welding of trunk pipelines up to 600 mm in diameter portable generators may have a capacity of 35 m³/hr. Machines from 5 m³/hr upwards are usually of the stationary type.

According to process control, all acetylene generators are divided into two general classes:

(a) those with *quantitative control*;
(b) those with *time control*.

In the former class, process control is effected by metering out either calcium carbide or water. Where the calcium carbide is the manipulated variable and the amount of water in the generating chamber remains constant, this is known as the carbide-to-water type. Where the water drops into the carbide, it is known as the water-to-carbide type. Wide use is also made of a combination type in which both the carbide and the water are metered out. This process is also utilised in generators producing dry slaked lime (the dry-residue type of generator).

In time-controlled generators, the carbide and the water are allowed to react at intervals. The quantity of acetylene produced per unit time (a minute or an hour) is controlled by varying the

duration of contact between the carbide and the water or the amount of carbide allowed to react with the water during each interval. If the carbide is manipulated for this purpose, this is known as the dipping type. Where the water is manipulated, this is known as the recession type. Time-controlled generators are also known as contact generators.

It is possible to combine the two types of process control in a single generator. As a result, a more positive control of the process is attained and the waste of gas into the atmosphere is reduced. It is advisable to combine the water-to-carbide type and the recession type, as has been embodied in ΓHB-1.25 generators developed by the Autogenous Welding Research Institute in the Soviet Union (see Sec. 3).

Fig. 9 shows diagrammatically the main types of acetylene generators available in the Soviet Union.

Among the advantages of carbide-to-water generators are a more complete reaction, better heat abstraction, better carbide utilisation, and well cooled and well purified acetylene. When necessary, carbide-to-water generators can be run at 150 per cent their rated capacity. A disadvantage of this generator type is increased water consumption for the reaction and gas cooling. For this reason carbide-to-water generators are bigger in size and produce much sludge. On top of all, they have to be equipped with involved mechanisms to control the amount of carbide discharged into the water tank. Because of this, carbide-to-water generators are mainly machines with an hourly capacity of over 20 cu m.

Water-to-carbide generators are simple in design, cheap to build, can use carbide widely differing in lump size, are convenient to operate, produce little sludge, and take little water. Their disadvantages are the possibility of overheating in the generating chamber and incomplete carbide decomposition. These generators cannot practically be run at higher than their capacity. These disadvantages grow in proportion with the size of generators, especially where as much as 30 to 50 kg or more carbide has to be loaded into the generating chamber. Their servicing is also handicapped. For this reason water-to-carbide generators are primarily of the portable type or small stationary machines producing not more than 20 cu m of acetylene an hour.

Recession-type generators are reliable in operation, convenient in servicing, and can be run at much higher than their rated capacity. Their principal drawback is that when a generator has been run at a high discharge rate, a sudden drop in gas consumption may expose much of the wetted carbide, the gas may be overheated and the sludge may produce a cake in the generating chamber. This drawback becomes more pronounced as the weight of the

carbide charge increases. Another disadvantage is that the heavier tray calls for the use of hoisting facilities. For this reason, recession-type generators are limited to capacities of up to 10 m³/hr.
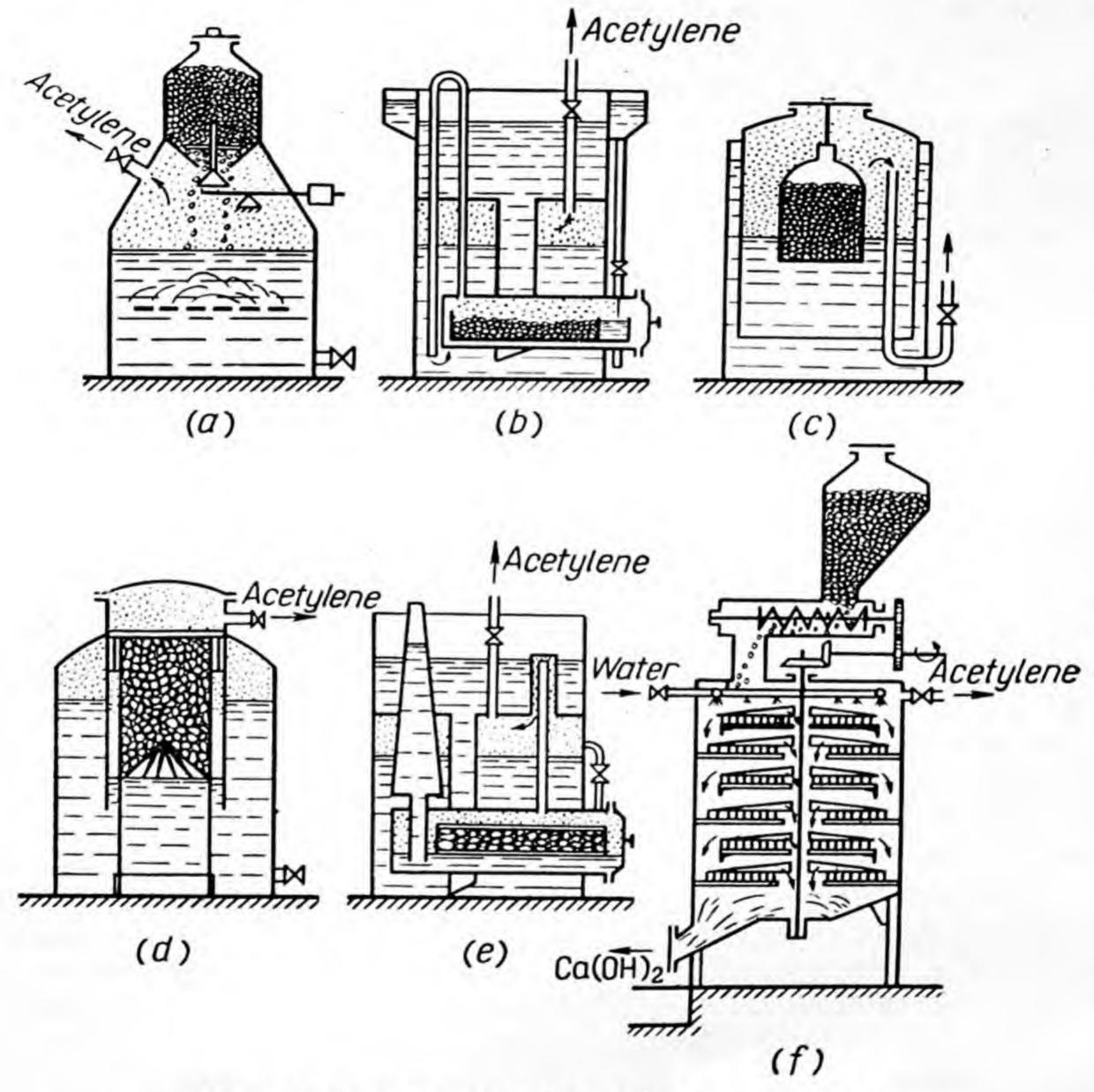
Fig. 9. Basic types of acetylene generators:

a — carbide-to-water;   *b* — water-to-carbide;   *c* — dipping   type;
d — recession   type;   *e* — combination   water-to-carbide   and   recession type;   *f* — dry-residue   type

In dipping-type generators the carbide is still more likely to be overheated in the generating chamber. Therefore, they are not safe to operate, and the use of this process should be discouraged even in small generators.

Combination water-to-carbide recession-type generators offer some of the advantages inherent in either class, while their demerits are less pronounced. The use of the recession process makes

it possible to reduce variations in the acetylene pressure in the generator when acetylene is withdrawn intermittently. The combination process has found use in generators up to 3.5 m³/hr in capacity.
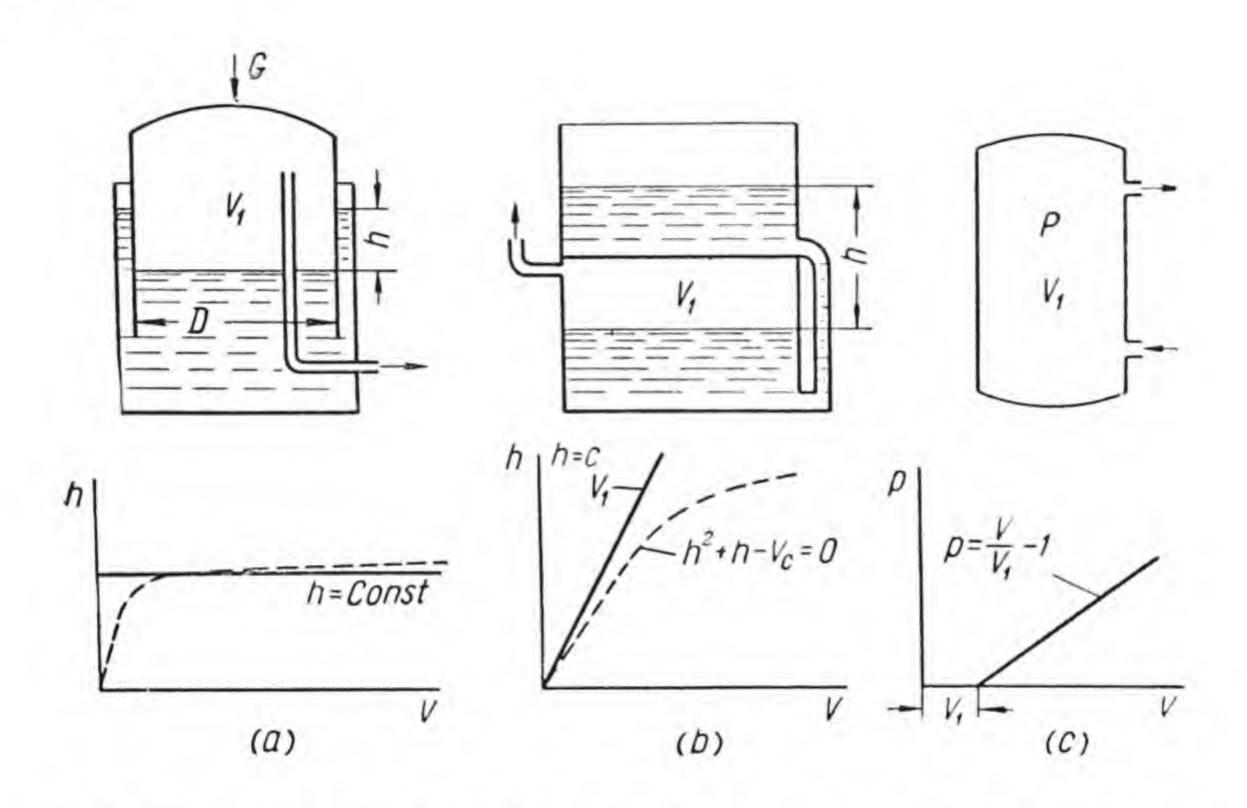
Fig. 10. Diagrammatic sketches of gas holders employed in acetylene generators

According to pressure, generators are classed into:

(a) low-pressure machines with a limiting pressure of up to 0.1 kg/cm² inclusive;

(b) medium-pressure machines, with a limiting pressure of over 0.1 kg/cm² to 1.5 kg/cm², inclusive;

(c) high-pressure machines with a limiting pressure of over 1.5 kg/cm².

The limiting pressure of a generator depends on the design of its gas holder.

There are three main types of gas holders (Fig. 10).

(1) A floating-bell (gasometer-type) gas holder (in Fig. 10a) is remarkable in that the gas pressure in it varies little with the discharge of gas from the holder.

Neglecting the loss of weight as the bell sinks in the water, as it is small in comparison with the weight of the bell, the gas pressure under the bell may be assumed to be constant and can

4*

be determined by the following equation:

$$p = \frac{G}{F \times 10^4} \text{ kg/cm}^2, \tag{III.1}$$

where p is the gas pressure in kg/cm²;

    G = the weight of the bell in kg;

    $F = 0.785D^2$ = the cross-sectional area of the bell in sq m.

As 1 kg/cm² = 10 m water column = 10,000 mm water column, then $h = G/F$, where h is the gas pressure in mm water column.

It is difficult to obtain high gas pressure with this type of gas holder, as this calls for a bell of an appreciable weight. Therefore bell-type gas holders are employed in installations producing acetylene under a low pressure of not higher than 400 or 500 mm w. g., or 0.04 to 0.05 kg/mm².

Fig. 10a relates the pressure of the gas in a bell-type gas holder (h) to the volume of this gas (V) at 20° C and 760 mm Hg. The dotted line takes into account the loss of weight by the bell as it sinks in the water and the variations in the volume of the gas with an increase in its pressure until the bell floats up.

Bell-type gas holders are employed only in stationary acetylene installations. Portable generators using this type of gas holder are not manufactured at all, as they are more dangerous to operate due to the possibility of air being sucked under the bell when there is little gas beneath it.

(2) *Communicating-vessel gas holders* (in Fig. 10b) are those in which the gas is held in the lower vessel and the water is squeezed out into the upper vessel. The limiting pressure depends on how high the upper vessel is located above the lower one. Denoting the volume of the gas holder by V_1 and the difference between the levels in the upper and lower vessels by h, then $h = CV_1$, where C is the proportionality factor.

The volume of gas, V, in a gas holder, under atmospheric pressure, is

$$V = V_1(h + 1). \tag{III.2}$$

Substituting V_1 in Eq. (III.2), we obtain:

$$V = \frac{1}{C}(h^2 + h) \text{ or } h^2 + h - CV = 0, \tag{III.3}$$

where h is the pressure in atm (gauge); 1 atm (gauge) — 10 m water column.

Fig. 10b plots V_1 and V as a function of h. In this type of gas holder, the pressure of the gas varies with and depends on the quantity of gas held. More often this type is employed in low-pres-

sure generators with a capacity of not more than 3 m³/hr, as a prohibitively large gas holder would be necessary to cater for a higher pressure or a greater capacity.

(3) A constant-volume gas holder (in Fig. 10c) is a vessel whose volume is independent of the quantity of gas in it. Denoting the volume of gas in the gas holder at 1 atm (abs) by V, the excess pressure of the gas in it by p and the volume of the gas holder by V_1, we obtain

$$(p+1)\,V_1 = V,$$

whence

$$p = \frac{V}{V_1} - 1. \tag{III.4}$$

Graphically, this relationship is a straight line with the co-ordinates p and V (Fig. 10c) intersecting the V axis at the point $V = V_1$.

As in the previous case, gas pressure in this type of gas holder is a variable. Therefore, regulators have to be used in order to be able to maintain the gas pressure constant as more gas is withdrawn from the holder. With this type of gas holder generators may be made compact in size, for which reason they are incorporated in medium- and high-pressure types.

2. Principal Requirements for Acetylene Generators

Irrespective of its design, every acetylene generator must meet the following principal requirements.

(1) The output of a generator must be sufficient to meet the demand for acetylene both in terms of quantity and flow rate.

(2) A generator must be designed to use a definite lump size (or sizes) of carbide.

(3) The decomposition of the carbide must be controlled automatically and in proportion to the gas consumption.

(4) A generator must have the highest possible coefficient of carbide utilisation. The coefficient of carbide utilisation is the ratio of the acetylene actually produced (V_a) to the acetylene the generation of which is guaranteed for a given grade of carbide by its supplier (V_c).

Example: $V_a = 2,500$ litres; $V_c = 280 \times 10 = 2,800$ litres, where 280 litres is the yield of acetylene per kilogram of carbide and 10 kg is the weight of the carbide charge in the generator. Hence, the coefficient of carbide utilisation is

$$\eta = \frac{2,500}{2,800} = 0.895.$$

Modern generators have a utilisation coefficient of 0.85 to 0.98.

(5) Provisions must be made in a generator for the best possible cooling of the carbide and water in the generating chamber and of the acetylene produced. The temperature of the water and sludge in the generating chamber must not exceed 80° C and that of the acetylene, 115° C. The temperature of the acetylene discharged from the generator into the pipeline or into the hose of a torch should not exceed the ambient temperature by more than 10 or 15° C.

(6) The pressure in a generator which has a gas-filled space should not exceed 1.5 atm (gauge).

(7) Provisions must be made in the design of a generator for the reliable blow-out of all gas spaces in the machine so as to remove all traces of air before it is filled with acetylene.

(8) A generator must be gas- and air-tight and have a gas holder of sufficient capacity so as to prevent the discharge of excess acetylene into the surrounding atmosphere in the case of a sudden drop in acetylene consumption.

(9) A portable generator must be as small as possible both in weight and in size.

(10) Provisions must be made in generator design for safety from explosion, for as full automatic control of carbide decomposition as practical, and for ample cooling and efficient purification of the acetylene.

3. Generator Design

Any acetylene generator, irrespective of its type, consists essentially of a generating chamber (or chambers), a gas holder, safety devices which prevent the pressure in the generator from rising too high and a water seal to prevent backfires to the generator. These parts may be built into a single unit or as separate units connected by pipes. In some cases stationary acetylene generators are equipped with acetylene purifiers which remove harmful impurities from the acetylene by a chemical method.

Here are some typical acetylene generators manufactured in the Soviet Union.

(a) *The Type ГНВ-1.25 Generator* (Fig. 11) is a portable machine of the combination water-to-carbide recession type. It consists of a shell *8* and a generating chamber *13*. The shell is divided into two halves by a partition *9*. The lower portion of the housing is connected with the generating chamber by a rubber hose *12* fitted with a cock *11*. Mounted on the shell is a water seal *2* connected with the gas space by a tube *7* and a rubber hose *6*. The carbide tray *14* is made from thin wire and is located inside the generating chamber at an angle. The carbide in the tray

is spread in a thin layer so that it can be efficiently washed by the water to prevent sludging.

The generator is filled with water to the desired level through the open top, with the cock *11* shut and the cock *5* open. The water seal is filled with water through a funnel *4* level with the pet
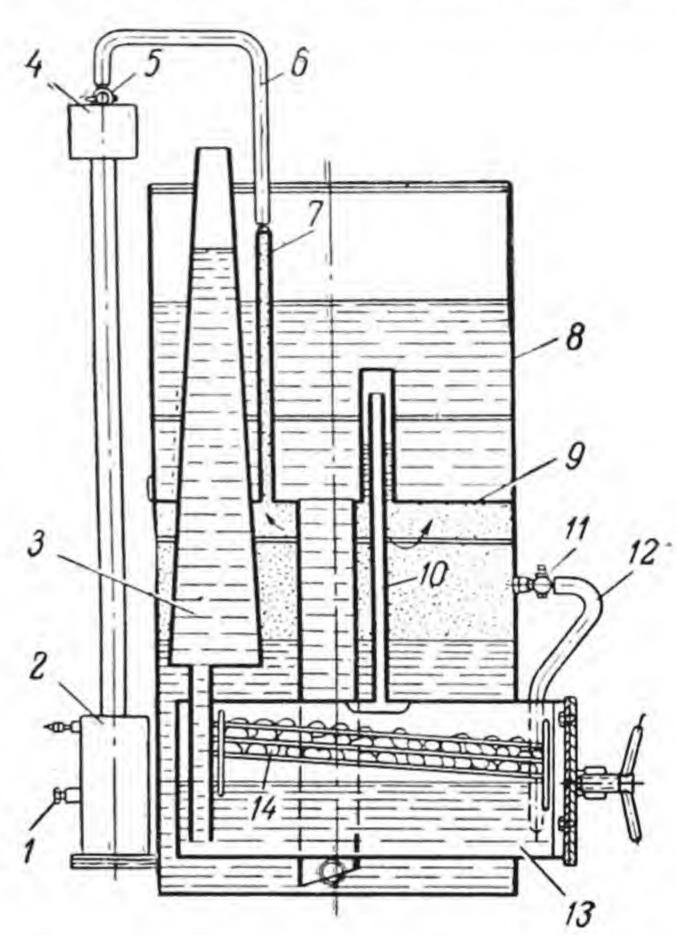
Fig. 11. Portable Type ГНВ-1.25 acetylene generator

cock *1*. The tray *14* is loaded with carbide and charged into the generating chamber which is then hermetically closed with a lid. The cock *11* is turned open, and the water enters the generating chamber from beneath, wetting the lower surface of the carbide lumps. The acetylene produced by the decomposition of the carbide leaves the generating chamber by a tube *10* and enters the generator shell, squeezing the water out of it into the open upper part. The water is fed into the generator until its level drops below the cock *11*. As more acetylene comes from the generating chamber, the pressure in the generator and chamber rises, and the water from the latter is squeezed out into a cone-shaped vessel *3* open at the top. As a result, the carbide in the tray is no longer wetted by the water, and the generation of acetylene stops. Owing to its conical shape, the pressure in the vessel *3* rises towards the end of the cycle quicker than at its beginning.

As acetylene is consumed, the pressure in the generator drops, the water from the vessel *3* fills the generating chamber again,

and the generation of acetylene is resumed. When the pressure in the generator drops to 230 or 250 mm (water column), the water in the lower part of the shell rises until it is level with the cock *11* and enters the generating chamber again by the hose *12.* The inflow of water to the generating chamber stops when the pressure rises to 250 or 260 mm (water column). The above action is automatic and depends entirely on the consumption of acetylene. The vessel *3*, its lower tube, the cock *11* and the hose *12* must be washed with water at least once a shift to prevent clogging by sludge.

The ГНВ-1.25 generator has a remarkably high coefficient of carbide utilisation, as it will not discharge any acetylene into the atmosphere even though there may be a sharp drop in gas consumption at a high flow rate. Instead, the pressure in the generator will only rise to 800 mm (water column). The generator delivers the gas at a sufficiently high pressure, which fact goes a long way towards making the operation of injector torches steady. A disadvantage of this generator is that it has only one generating chamber, and welding or cutting has to be stopped each time the chamber has to be recharged.

(b) The Type МГ-55 Generator (Fig. 12) is a portable machine of the water-to-carbide type. It is a modernised version of the МГ-44. It consists of a cylindrical shell *3* open at the top and divided into two parts by a horizontal partition *4.* The lower part serves as a gas holder and the upper part as a water tank. Both parts are connected by a circulation pipe *8.* Into the lower part are welded two cylindrical generating chambers *2* which receive carbide trays *1.* Each tray is divided by vertical partitions into six compartments which are filled with calcium carbide.

The generating chambers are hermetically closed by lids *11* which are fitted with rubber gaskets. Water to the generating chambers is admitted from an annular reservoir *7* through infeed pipe *9* and two siphon tubes *12* fitted with cocks *10.* The flow of water into the generating chambers is controlled automatically and depends on the gas pressure in it and, consequently, on the consumption of acetylene.
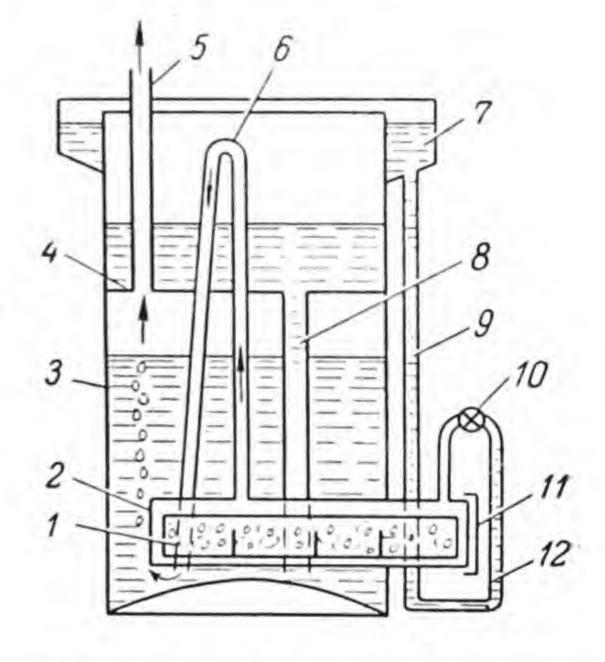
When no acetylene is withdrawn from the generator, the flow of water to the generating chamber is stopped automatically, and so is the generation of acetylene. The gas from the generating chambers is withdrawn by the gas tubes *6* to the gas holder. The acetylene squeezes the water out of the gas holder into the water tank by the circulation pipe *8.* As acetylene is consumed, the gas holder is filled with water from the water tank. Gradually, as more acetylene is withdrawn from the generator, all compartments in the tray and the respective generating chamber are filled with

water. The inflow of water to the chamber with the decomposed carbide is then cut off, and water is directed through the respective cock *10* to the other generating chamber. Leaving the gas holder by the pipe *5*, the acetylene enters the water seal whence it is delivered to the welding torch.

Carbide utilisation in МГ-44 generators, some of which are still in service, can be raised by replacing the shorter leg of the siphon pipe with a new one made from a pipe 3/4-inch in diameter

Fig. 12. Diagram of Type МГ-55 acetylene generator

and 100 mm long (as measured from the top of the generating chamber) and by reducing the effective orifice of the water cocks on the generating chambers down to 4 mm.

Table 6 summarises the performance of the portable acetylene generators manufactured in the Soviet Union.

(c) The Type ГРК-10 Generator (Fig. 13) developed by the Autogenous Welding Research Institute is a stationary machine of the water-to-carbide type. It has two horizontal cylindrical gas holders *1* and *3* and two generating chambers *12*. The generating chambers are enclosed in a box *5* filled with running water for better cooling of the generating chambers. Each generating chamber receives two carbide trays *11*. Water is admitted into the generating chambers via a spring-loaded diaphragm regulator *9*. When the gas pressure in the chambers rises to 0.9 or 0.95 atm (gauge), the regulator automatically cuts off the inflow of water. As soon as the pressure drops to 0.85 or 0.9 atm (gauge), water is again admitted to the generating chambers. The acetylene given up by the carbide ascends the pipe *7*, passes through the nonreturn valve *6* and enters the lower gas holder *3* where it is scrubbed with water and is delivered to the line via a pressure

Table 6

Performance of Portable Acetylene Generators

Characteristic	Type designation						
	ГВД-0.8	МГВ-0.8	МГ-55	ГНВ-1.25	ГВР-1.25М	ГВР-3	ГПГ-35
Type	Recession	Recession	Water-to-carbide	Combination water-to-carbide/recession			Carbide-to-water
Normal capacity, m³/hr	0.8	0.8	2.0	1.25	1.25	3.0	35
Working acetylene pressure, atm (gauge)	0.07-0.3	0.08-0.3	350 mm w. g.	160-240 mm w. g.	0.08-0.15	0.15-0.30	0.9-1.2
Maximum pressure in generator, atm (gauge)	0.3	1.5	900 mm w. g.	800 mm w. g.	0.7	0.7	1.4
Generator height, mm	590	780	1,135	1,040	1,042	1,260	2,325
Shell diameter, mm	260	290	590	478	480	630	1,550
Water requirements, litres:							
(a) to fill shell	10	10	180	85	38	60	1,400
(b) to fill tank	—	—	23	—	19	35	—
Water per kg CaO₂, litres	5	5	3.2	4.2	4.2	3.8	8-9
Carbide charge, kg	2	2	2×2.5	1×4	1×4	2×4	150
Carbide lump size, mm	25/50; 50/80	25/50; 50/80	15/25; 25/50	25/50; 50/80	25/50; 50/80	25/50; 50/80	2/8
Carbide utilisation	—	—	0.80-0.90	0.94	0.86	0.86	0.96
Total weight, less water and carbide, kg	19.5	19	65	42	54	110	750

regulator *8* and the respective water seal *2*. The generator has a panel *4* where the pressure gauges and water cocks are arranged.

After the carbide in one of the chambers has been fully decomposed the water is automatically diverted through the respective cock to the other chamber already charged with carbide.
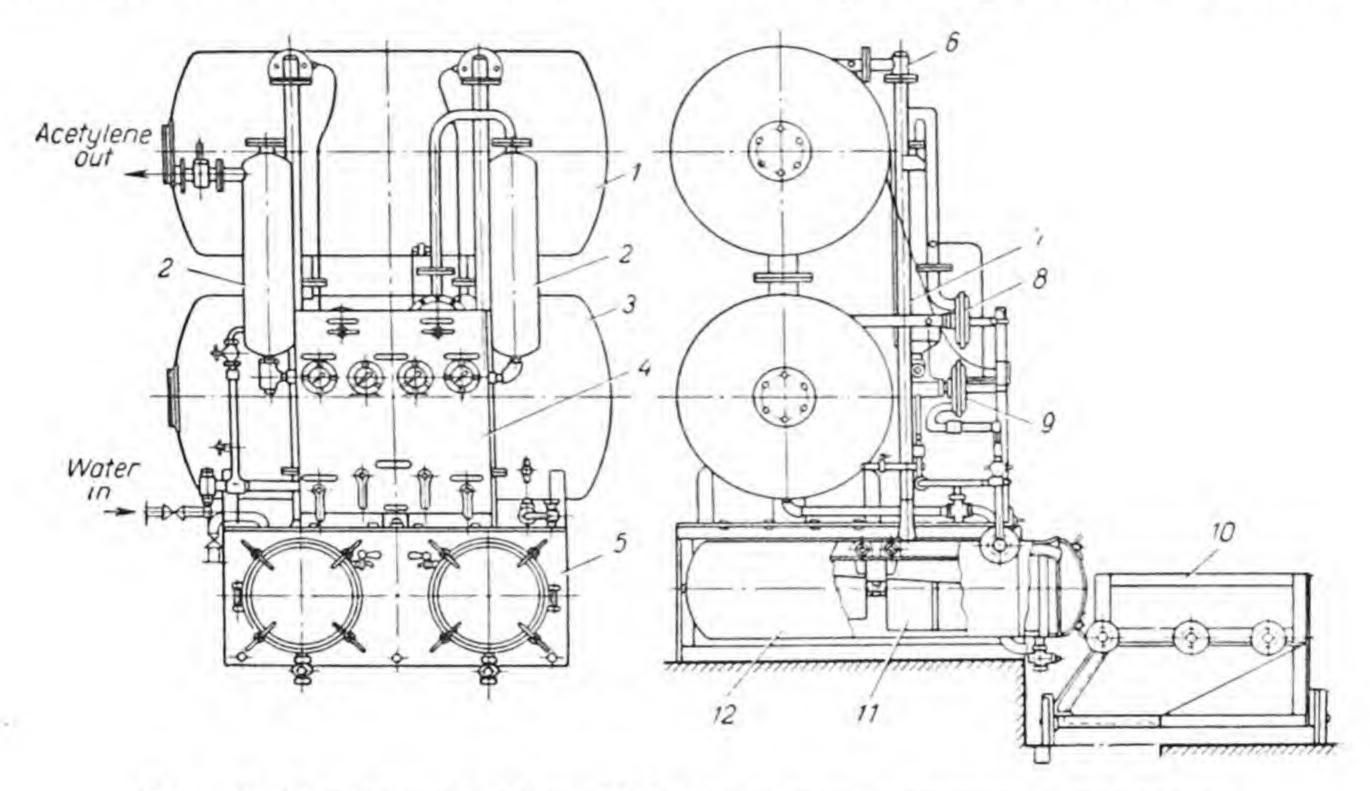
Fig. 13. Diagram of Type ГРК-10 stationary acetylene generator

After the other chamber has been in operation for 8 or 10 minutes, the first chamber is disconnected and opened for flushing and charging with a fresh load of carbide. The carbide trays removed from the generating chambers are transferred onto a trolley *10* on which they are flushed with water.

4. Generators Using Carbide Smalls and Dust

Carbide smalls and dust decompose almost instantaneously upon contact with water. In conventional generators the heat given up by the decomposition of carbide smalls and dust cannot be abstracted by the cooling water as quickly as necessary.

As a result, the sludge, acetylene and undecomposed carbide dust are heated to 400° C or even higher. Under increased pressure or in the presence of air in the acetylene, this temperature rise is fraught with an explosion of the acetylene-air mixture. Therefore, it is good practice, when using carbide dust, to have an excess of water in the generating chamber and to stir it vigorously, especially at the interface with the carbide. This

technique helps to remove sludge foam and to improve the reaction between the carbide and the water.

This principle is embodied in acetylene generators using carbide dust (Fig. 14). Cirbide dust is charged into a hopper *1* from which it is fed via a valve *2* into a lower hopper *3*. An auger *4* feeds the dust into the shaft of the generator where it is discharged into the water and is stirred vigorously by paddles *7*. A continuous flow of water is admitted into a reservoir *5* so that it
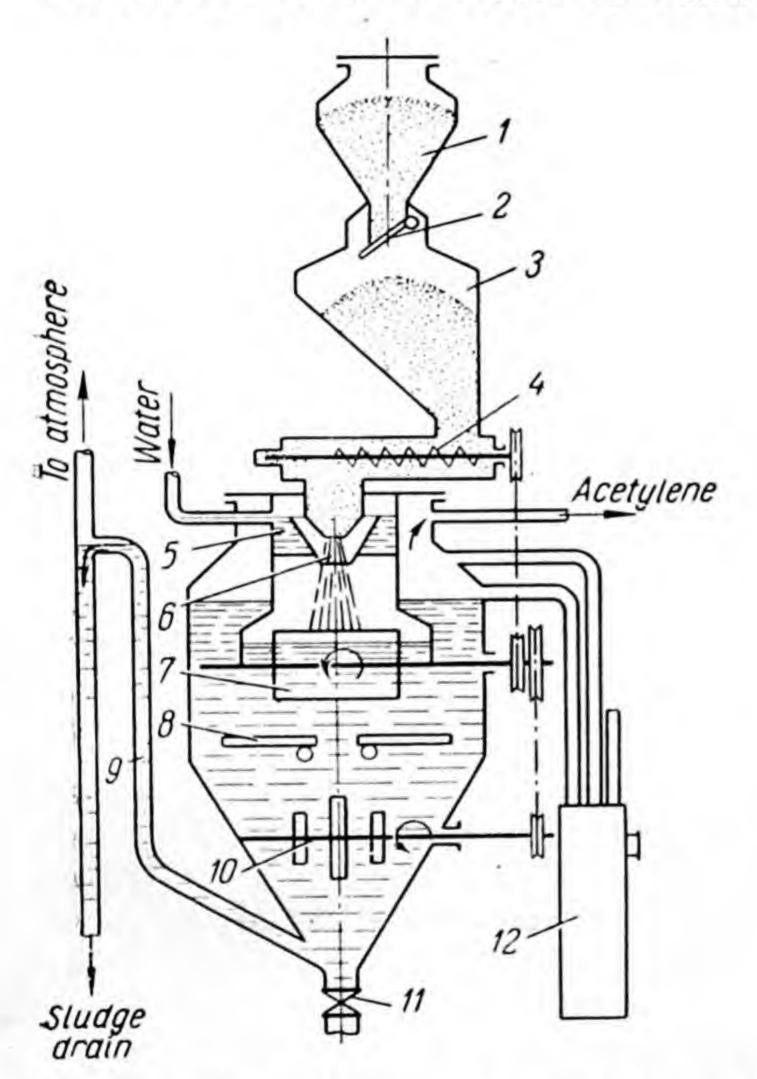
Fig. 14. Diagram of an acetylene generator using carbide dust and smalls

flows over a funnel *6*, washing down specks of carbide dust falling out of the auger. The larger particles and lumps of ferrosilicon are trapped by grates *8*. The liquid sludge is continuously decanted through a siphon tube *9*. The thicker sludge and ferrosilicon are drained through a cock *11* at regular intervals. The sludge is stirred by a stirrer *10*. The water seal *12* discharges excess water and prevents the pressure in the generator from rising suddenly. Surplus gas is discharged into the gas holder.

Fig. 15 shows diagrammatically the Type ГМК-10 generator developed by the Autogenous Welding Research Institute for operation on carbide smalls. When carbide is crushed some 15 to 20

per cent of it comes out as smalls 2/8 and 8/15 mm in size. These smalls can be used in Type ГМК-10 generators to produce acetylene for flame metal-working processes.

The ГМК-10 generator is a stationary medium-pressure machine of the carbide-to-water type. The generator shell is a vertical welded cylinder *1*. The feeding mechanism incorporating an auger *3* which feeds carbide into the generator and a diaphragm-type pneumatic drive *7* which rotates the auger are mounted on
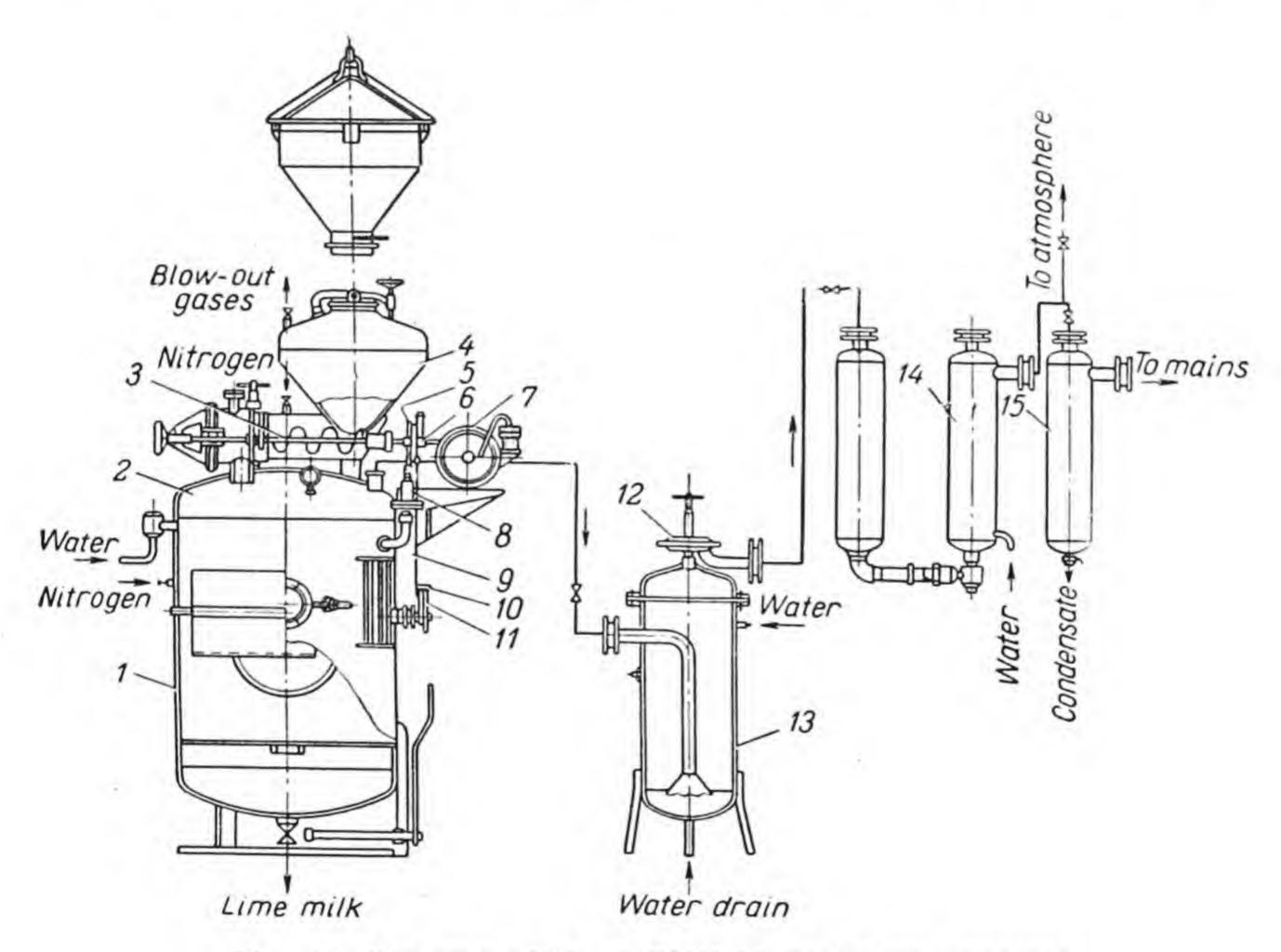
Fig. 15. Diagram of Type ГМК-10 acetylene generator

the top *2* of the shell. Inside the shell there is a grate on which the carbide is decomposed. Mounted on a shaft above the grate is a stirrer driven by the diaphragm-type pneumatic drive *7* via a link rod *9*, a ratchet wheel *11* and a pawl *10*. The feeding mechanism is started automatically by a diaphragm *8*. The generator is connected by a pipeline with a scrubber *13* which has a pressure regulator *12* to control the acetylene pressure in the line to the torch. From the regulator the gas is directed to a water seal *14*, past a drip pot *15*, and into the line.

The feeding mechanism incorporates a cone-shaped hopper *4* and an auger *3* rotating on a shaft. The shaft extension carries a ratchet wheel *5* which engages a pawl *6*. The auger trough is connected with the hopper *4*.

The diaphragm-type pneumatic drive *7* consists of a housing divided into two compartments and two diaphragms facing inwards and closing the two compartments. The diaphragms bear upon the connecting rod of the shaft. A slide valve admits compressed air alternately in either compartment, the diaphragms impart an oscillating movement to the shaft, and the ratchet wheel and pawl convert it into a rotary movement.

The feeding mechanism is automatically started and stopped by the diaphragm *8* and a system of linkages which cause the pawl to engage or disengage the ratchet wheel on the auger shaft, as the pressure in the generator varies.

5. Dry-residue Generators

The dry-residue process uses a small excess of water which is sprayed before it is allowed to react with the carbide. The heat of the reaction goes to evaporate this water. The hydrated lime produced by the reaction is a dry powder.

The dry process uses 1 to 1.2 litres of water per kilogram of carbide. Out of this amount 0.56 litre goes to decompose the carbide, and the balance is evaporated. The process proceeds at 90 to 110° C. This process has the following advantages over the processes based on the decomposition of carbide in an excess of water:

(1) improved carbide utilisation, as there is no liquid sludge to carry away any acetylene with it;

(2) generators can be constructed to produce as much as 1,000 to 2,000 cu m of acetylene an hour;

(3) no sludge to store and dispose of;

(4) hydrated lime can be used in carbide production, in building mortar, etc.

The disadvantages of the dry process are:

(1) generator design is of necessity more complicated, as water and carbide have to be metered out in precise amounts, and their feed has therefore to be automated; otherwise, the temperature in the generating chamber may rise suddenly, should either the carbide or water be fed in amounts other than specified;

(2) it is difficult to feed small amounts of water in a well dispersed condition reliably and continuously, as the water nozzles are small and can be easily clogged by lime dust;

(3) generators of high capacity have to use fine-sized car-
bide lumps, which necessitates the additional disintegration of
large carbide lumps;

(4) the resultant acetylene carries increased amounts of sul-
phuric and phosphoric impurities, ammonia and lime dust, which
necessitates the provision of additional facilities for scrubbing,
cooling and purifying the acetylene.

Fig. 9*f* shows diagrammatically the shelf type of dry-residue
generator.

Dry-residue generators may also be of the drum type and of
the combination shelf-drum type.

In drum-type dry-residue generators the carbide is decomposed
inside a rotating inclined drum made from a wire net. The
carbide lumps are continuously shifted across the drum, rubbing
against, and thus removing the envelope of hydrated lime from,
one another.

6. Automatic Acetylene Generator Installations
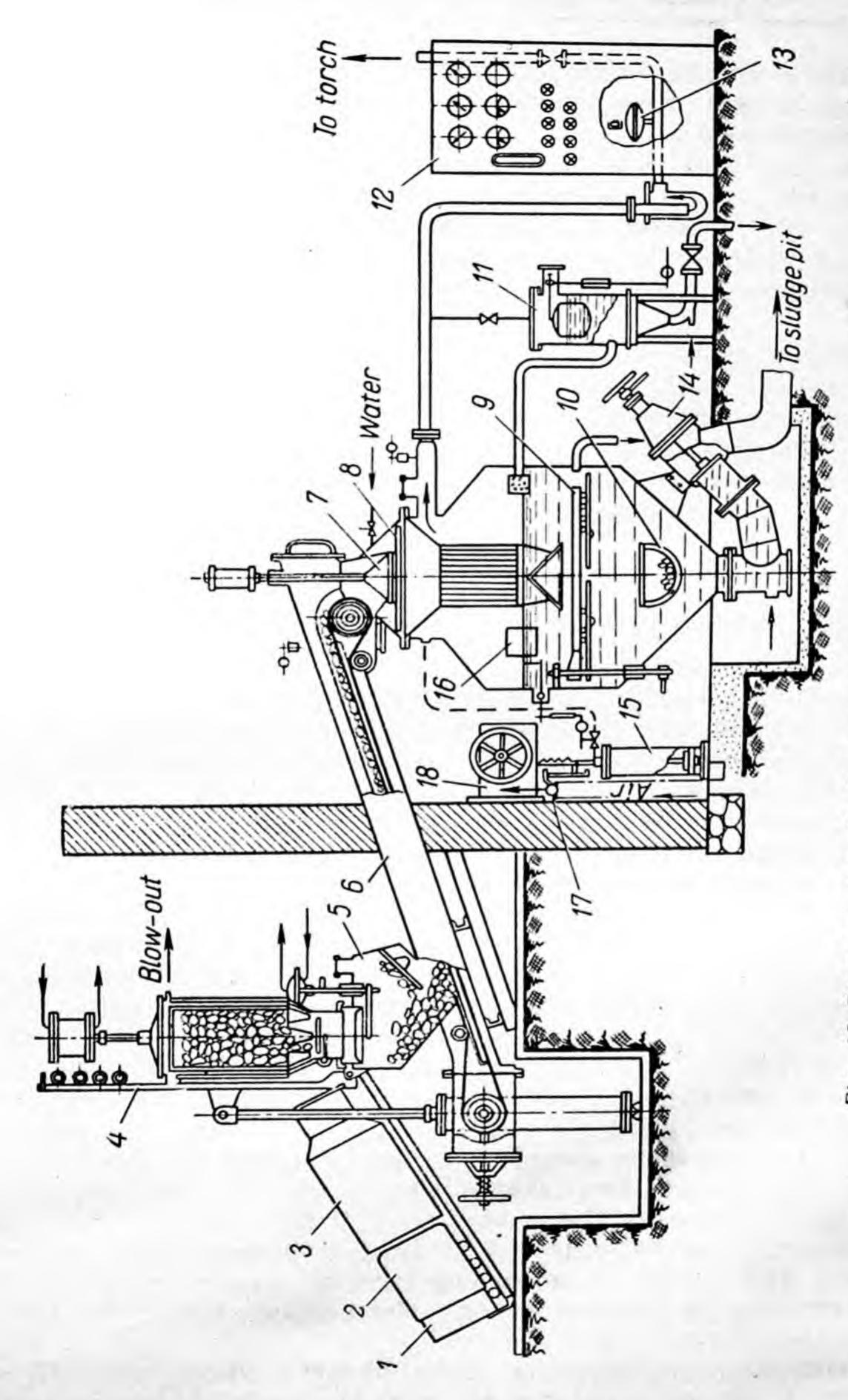
Modern stationary installations use mechanised and automat-
ed acetylene generators. A suitable example is provided by the
Avtogend-M generator designed by Y. V. Dalago (Fig. 16).

This is a medium-pressure machine operating continuously on
any size of carbide lumps (including dust). Carbide is charged
into the generator by an air-operated loader *4* from a storage
drum *1*. For this purpose, the bottom of the drum is cut away and
a hood *2* is put on instead. The hood automatically opens when
the drum enters the casing *3* of the air-operated loader. The lat-
ter turns and places the drum on the neck of a hopper *5*. From
the hopper the carbide is transported by an enclosed conveyer *6*
into the generator. The conveyer is started automatically as the
pressure of the acetylene in the generator drops to below the set
limit. At that moment a valve *13* automatically admits compressed
air into an air cylinder *15* whose piston moves the rack, the
rack actuates the pinion *18* and, through the latter, the pulley of
the chain transmission coupled to the driving sprocket of the
conveyer. The size of the carbide charge can be adjusted at will
by varying the travel of the rack.

As the conveyer is put into motion, a cone-shaped shutter *7*
is opened, and water is admitted into a sprayer *8* to wet the
dust which may be present among the carbide lumps discharged
into the generator.

The charging operation is repeated by a slide valve *17* and
the piston of the air cylinder *15* until the carbide charge is suf-
ficient to build up the necessary pressure in the generator.

Fig. 16. Diagram of Y. V. Dalago's Avtogend-M acetylene generator

When the pressure reaches the desired limit, the valve *13* shuts off compressed air until the acetylene pressure is again below the minimum limit. Usually the difference between the maximum and minimum pressure limits is 0.1 to 0.12 atm (gauge).

The ferrosilicon and other insoluble impurities are dumped from a grate *9* into a suspended basket *10* which can be removed through a separate hole when the generating chamber has to be cleaned.

Although it has only one generating chamber, the generator can operate continuously for two or three weeks. This is due to the fact that the temperature in the generator is maintained practically constant by float-and-valve systems *16* and *11* which automatically admit cold and discharge hot water.

The sludge is removed from the generating chamber without stopping it through a valve *14*, a hermetically sealed drain and into sludge pits.

Provisions are made in the generator for blowing it out with carbon dioxide and nitrogen and for discharging their mixture into the atmosphere.

The generator is run remotely and all the controls, including those for manual operation, are concentrated on a control board *12*.

The generator has bursting diaphragms and lever-type safety valves mounted separately on the feeding mechanism and the generating chamber.

That the generating chamber can be disconnected from the system and carbide charged by a cone-shaped shutter *7* enhances safety to personnel and makes it possible to keep the carbide on the conveyer belt even during prolonged interruptions in generator operation. Thus, the generator can be started at a moment's notice.

That the temperature of the water in the generating chamber may be raised to as high as 70 or 75° C adds to the safety * and reduces the loss of acetylene with the sludge.

The generator is compact and small in size; the height to which the carbide charge has to be lifted is reduced to a minimum; there are no difficulties involved in the provision of servicing platforms; and the floor space of the acetylene generator house can be reduced by as much as 40 per cent, as compared with other generators.

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* When the temperature in the generator rises, the content of water vapour in the acetylene increases, and the explosibility of the gas is reduced.

The technical data on the Avtogend-M generator are given below.

Normal capacity, m³/hr	Up to 50
Working pressure in generator, kg/cm²	Up to 0.7
Carbide lump size	Any spesified by standard
Generator height, m	2.7
Diameter of generating chamber, m	1.5
Water consumption, litres/kg	7
Air consumption, litres/kg	20
Carbon dioxide consumption, kg/kg	0.02
Carbide utilisation, per cent	99
Weight of generator, kg	1,300

7. Water Safety Seals

(1) Flashback. The safety of using an oxy-acetylene flame for heating or melting metals is dependent on the stability of the flame. This in turn depends on the conditions under which the mixture burns as it issues from the tip of the torch, or on the relationship between the rate of flame propagation, or flame velocity, and the velocity of the mixture.

As it emerges from the tip of the torch, the oxy-acetylene mixture catches fire, producing a flame. The lower annular portion of the flame, next to the tip, is a continuous annular source of ignition. The inner cone of the flame is the front of a steadily burning oxy-acetylene flame.

Fig. 17 shows diagrammatically a cross-section through the front, or the inner cone, of an oxy-acetylene flame. The unit area df of this flame front continuously receives the fuel mixture from an outlet port with a radius r at a constant velocity w. At the same time, the unit area df, where the mixture is ignited, tends to move at a right angle to the flame front at a speed u.

The linear velocity of an element of the flame front towards its surface at a given point is called the normal flame velocity, u. For an oxy-acetylene flame to burn steadily it is essential that

$$u = w \sin \alpha, \qquad\qquad (III.5)$$

where α is the angle between the vector of gas velocity w (the axis of the gas flow) and a tangent to the surface of the flame front at a given point.

Let us denote the projections of the unit area df of the flame front on the respective co-ordinate planes by $d\rho$ and dz (Fig. 17).

Then:

$$\sqrt{(dz)^2 + (d\rho)^2} \sin \alpha = - d\rho;$$
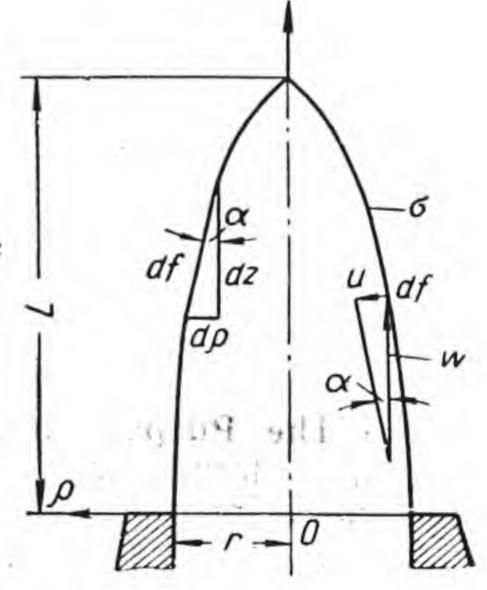
$$dz = \pm \sqrt{\left(\frac{1}{\sin^2 \alpha} - 1\right)} d\rho.$$

Substituting the value of $\sin \alpha$ from Eq. (III.5), we obtain

$$dz = \pm \sqrt{\left(\frac{w}{u}\right)^2 - 1}\, d\rho \tag{III.6}$$

From Eq. (III.6) it follows that the length L of the cone (Fig. 17) will be the longer, the greater is the ratio w/u. At some

Fig. 17. A cross-section through the inner cone of the oxy-acetylene flame

limiting value of w/u the flame will *break away* from the tip. As the ratio w/u decreases, the length L of the cone decreases too, and at $w/u = 1$ it will be zero. The flame front will become flat, and the gas will burn inside the torch. This is the beginning of a *flashback*.

Thus, a flashback is the return of the flame into the torch and its propagation against the flow of the oxy-acetylene mixture. In the final analysis, the possibility of a flashback depends on the relationship between the velocity, w, at which the gas mixture issues from the tip and the normal flame velocity, u. Its possibility increases with decreasing w, increasing u, and with the inner cone L growing shorter.

The velocity w decreases with increasing radius r of the tip orifice and with decreasing pressure and mixture flow rate. The normal flame velocity, u, increases with increasing gas temperature and oxygen content in the mixture until the flame velocity

is the highest. For this reason, these parameters, too, have a decisive effect on the possibility of a flashback.

Fig. 18 relates the average velocity of the mixture at which a flashback occurs and the temperature of the gas mixture in the tip. By average velocity is meant the ratio of the volume flow rate of the mixture per second to the area of the tip orifice. When the temperature of the gas mixture is raised to 250° C, the flame velocity sharply increases, and a flashback may occur at a higher average velocity. An increased oxygen content in the gas
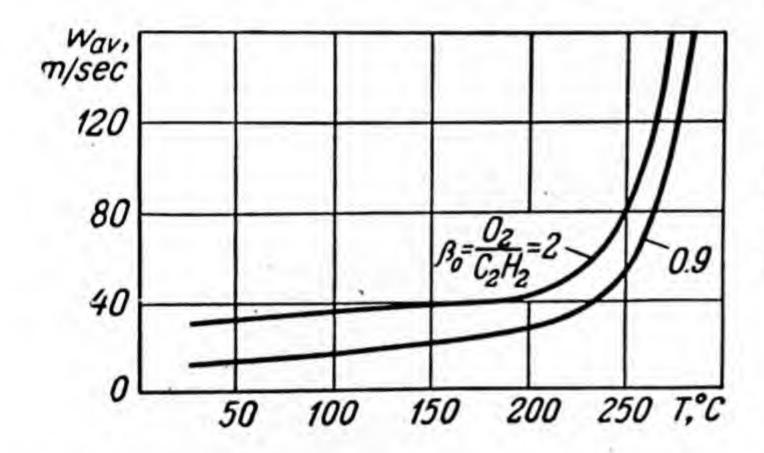
Fig. 18. Average velocity of the oxyacetylene mixture at which a flashback occurs as plotted against the temperature and composition of the mixture in the tip

mixture also raises the lower limit of the average velocity of the mixture at which a flashback is likely to take place.

(2) The Purpose and Classification of Water Seals. A water seal is a device which prevents a flashback or a reverse flow of oxygen from the torch to the gas line or the acetylene generator. In the case of low-pressure generators it also prevents the formation of a vacuum or the suction of atmospheric air into them.

An efficient water seal must meet the following principal requirements:

(a) it should reliably block the way for a flashback or the shock wave and discharge the explosive mixture into the atmosphere;

(b) it should be strong enough to withstand the pressure built up by an explosion of the fuel mixture;

(c) it should offer least hydraulic resistance to the flow of gas;

(d) it should lose as little water to the stream of gas as possible;

(e) it should be easily accessible for inspection, washing and repair.

Existing designs of water seals can be classed in a variety of ways:

(a) they can be classed by design into low-pressure (or open) and medium-pressure (closed) types. Open water seals can be of the station variety for pressures not over 1 m w. g. and of the manifold variety for pressures up to 2.5 or 3 m w. g. Closed water seals are manufactured for a limiting pressure of 0.7 to 1.5 atm (gauge);

(b) they can be classed by throughput into station water seals capable of handling up to 3.2 cu m of acetylene per hour and master or manifold water seals designed to handle 5 to 10, 20 to 35, and 50 to 75 cu m of acetylene per hour.

(3) Open Water Seals. The construction and operating principle of an open water seal are shown diagrammatically in
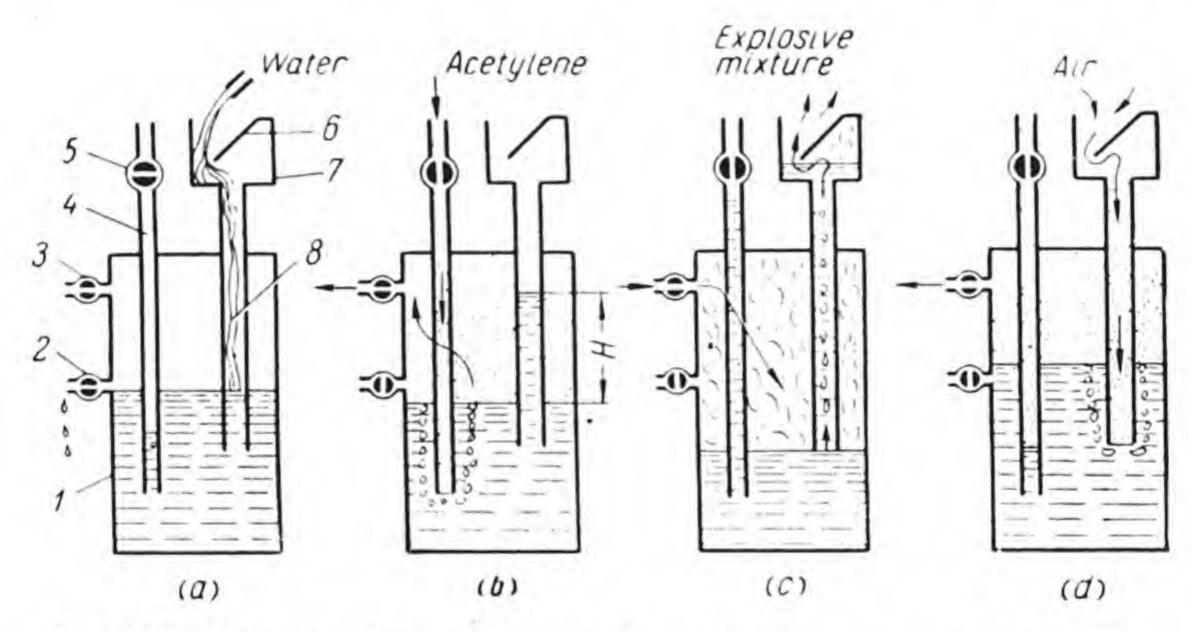
Fig. 19. Construction and operation of an open-type (low-pressure) water seal:

a — seal being filled with water; *b* — straight flow of gas through seal; *c* — flashback; *d* — suction of air through seal when too little gas is present in generator or gas line

Fig. 19. This is a low-pressure water seal consisting of a housing *1*, a gas inlet tube *4* and a safety tube *8* which is somewhat shorter than the gas inlet tube and has a funnel *7* and a baffle *6* at the top. The gas is discharged from the water seal through a cock *3*. The water level in the seal is checked by means of another cock *2*. A third cock *5* serves to disconnect the water seal from the line. The upper surface of the water in the safety tube is open to the atmosphere. The water column *H* in this tube (Fig. 19*b*) is balanced by the excess pressure of the gas in the seal.

Fig. 20 shows two typical designs of low-pressure station wa-
ter seals. The seals can be taken apart for inspection of their
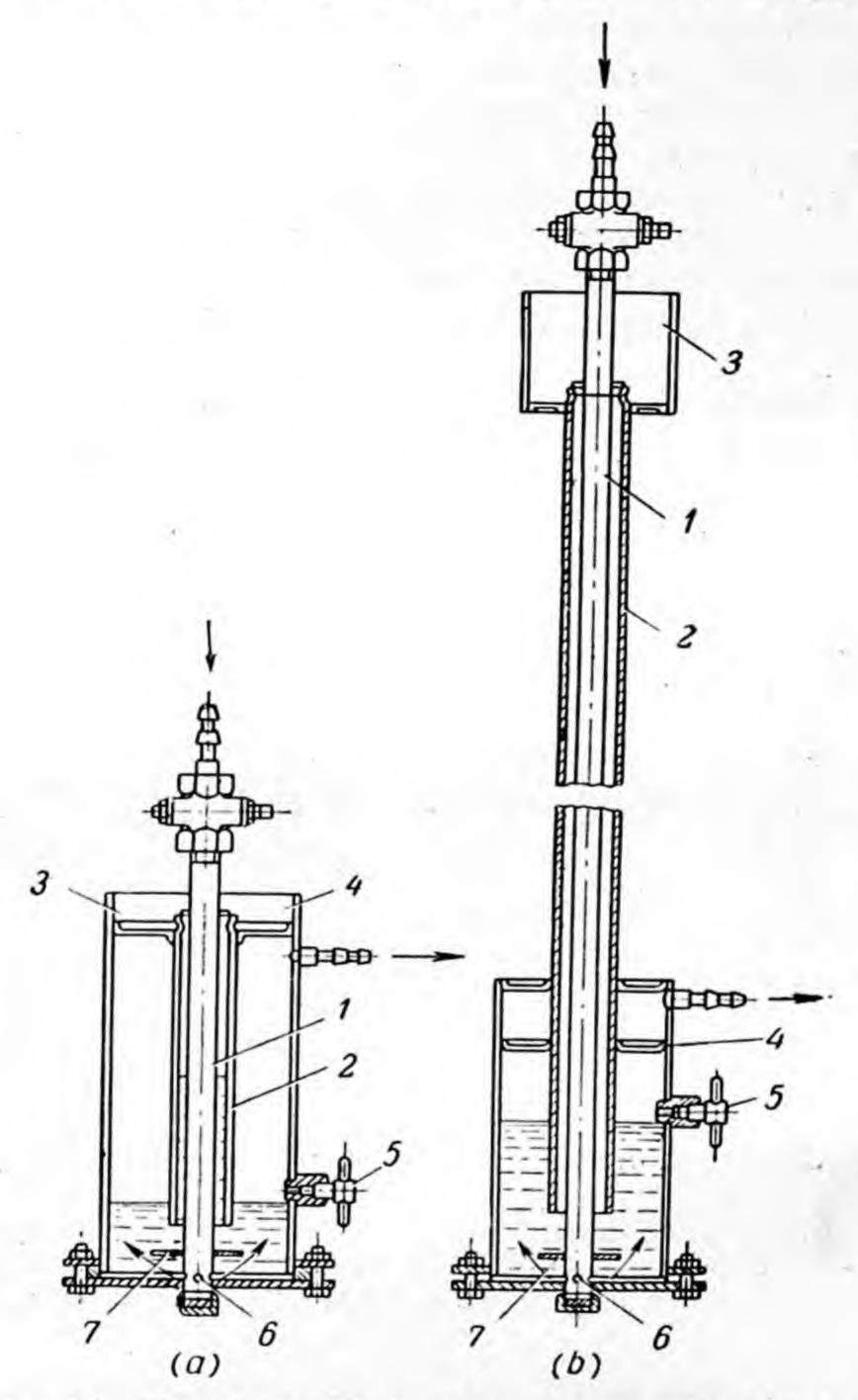
Fig. 20. Diagrams of low-pressure station water
seals:
a — for 1.5 m³/hr and a pressure of up to 200 mm
w. g.; b — for 3.2 m³/hr and a pressure of up to
1,000 mm w. g.

internal parts. The gas inlet tube *1* is placed inside the safety
tube *2*. Issuing from the holes *6* at the bottom end of the inner
tube, the gas stream is broken up into separate jets by a disk *7*
located above the holes. This reduces the possibility of a single

gas stream forming inside the seal. Otherwise, the shock wave produced by a flashback would be able to propagate through this stream and into the gas inlet tube. The seals have water baffles *4* located in the gas spaces. These baffles catch the drops of water entrapped by the stream of gas. The seals are filled with water through the funnels *3* level with the pet cocks *5*.

Fig. 21 shows diagrammatically the design and basic dimensions of a manifold water seal with a throughput of 35 m³/hr
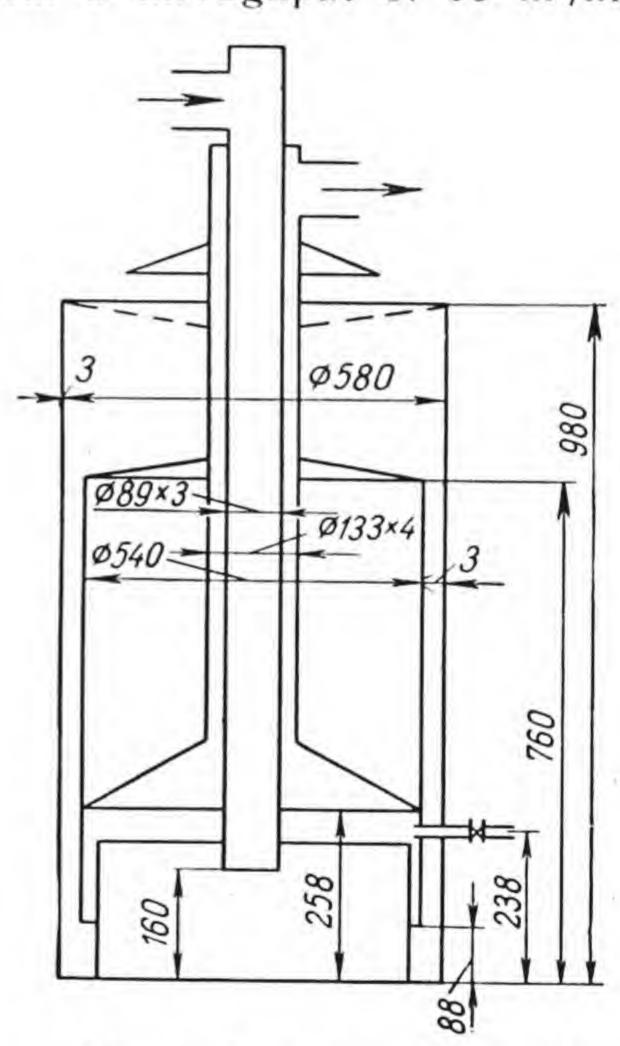
Fig. 21. Diagram of a manifold water seal for low pressure and a flow rate of 35 m³/hr

and a pressure of up to 500 mm. w. g. The technical data on station seals are summarised in Table 7.

(4) Closed Water Seals. The design and operating principle of a closed water seal are shown diagrammatically in Fig. 22. The seal consists of a housing *1* filled with water level with a pet cock *8*. The tube *2* admits the gas which passes through a valve *3*, a splitting disk *4*, a column of water and into a drip catcher *5* whence it flows through a pipe union *6* to enter the hose. At the top the tube has an aluminium-foil diaphragm *7*. When a flashback occurs, the diaphragm bursts open to let the explosive mixture out into the atmosphere. The pressure produced by the

Table 7

Technical Data and Basic Dimensions of Open Low-pressure Station Water Seals

Characteristic	Station water seal (Fig. 20)	
	Type *a*	Type *b*
Flow, m³/hr	Up to 1.5	Up to 3.2
Maximum pressure, m w. g.	0.2	1.0
Resistance at maximum flow, m w. g. . .	0.185	0.45
Velocity in gas inlet tube, m/sec	0.022	0.044
Water volume, litres	0.4	1.3
Gas space, litres	1.1	1.0
Structural dimensions, mm:		
D	89.5	109.5
b	1.25	1.25
H	385	250
h	64	140
Δh	52	55
Δh'	12	85
O. D. and wall thickness of gas inlet tube, mm	17 ×2.25	21.25 ×2.75
Same, safety tube, mm	28×2.0	35×1.5
Diameter and number of holes for gas discharge, mm/holes	6/6	5/6

explosion in the seal is transmitted through the water to the valve *3* which closes to disconnect the gas line from the torch.

The shock wave can travel past such a water seal only in two cases, namely (a) when the disk fails to break up the gas stream efficiently and a foam of gas-liquid emulsion is formed within the seal; and (b) when the nonreturn valve fails to respond quickly to the pressure rise brought about by the explosion. Therefore, the principal points to watch in closed water seals are the velocity of the gas through the seal and the design of the nonreturn valve and stream splitter.

Closed water seals are used on pressures of over 0.1 atm (gauge) and are very compact and reliable in operation. Fig. 23

shows (a) a station and (b) a manifold water seal of the closed type. The valves *1* of the seals are steel balls moulded in soft elastic rubber. Alternatively, the valves may be brass cones. The gas distributors *2* are perforated caps which limit the lift of the valves when the gas flows through them. The seal at "*b*" has an additional gas distributor in the form of a gauze. The pet cocks *3* are located so that there is a sufficient column of water above
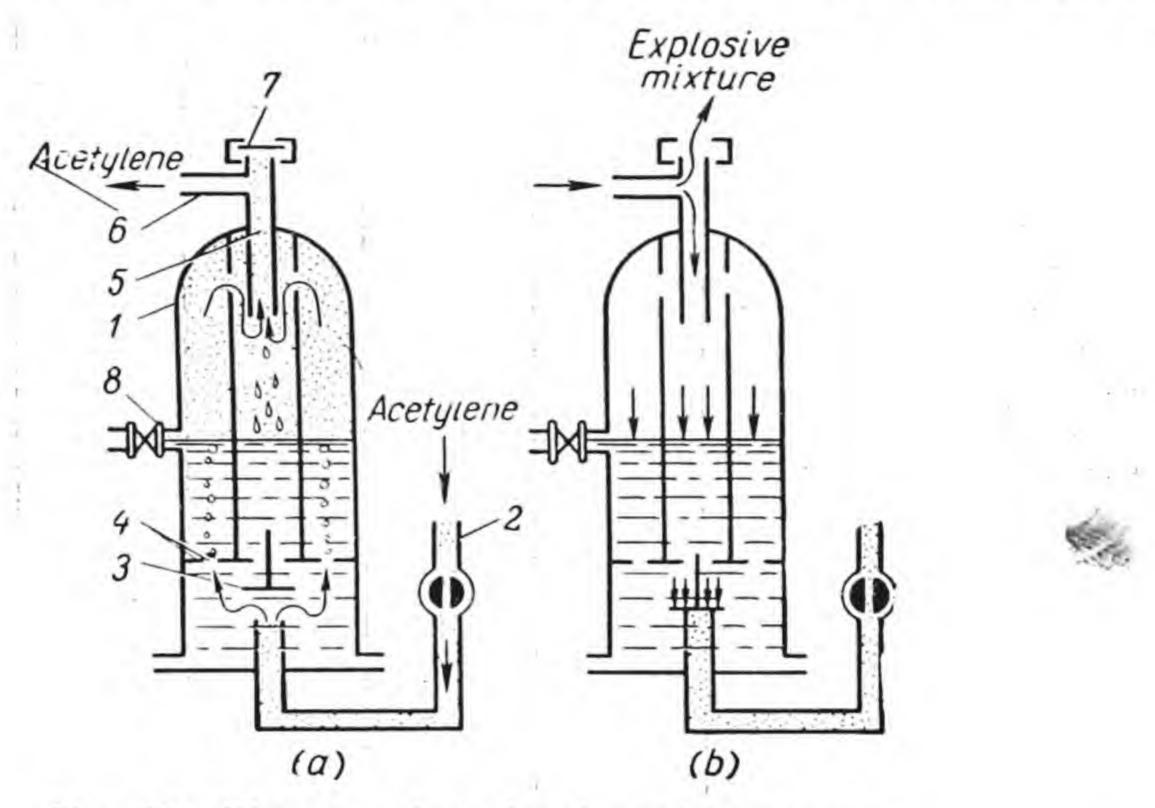
Fig. 22. Diagram of a closed medium-pressure
water seal:
a — normal operation; *b* — flashback

the valve and water drops can be separated efficiently in the gas space of the seal. The diaphragms *4* are tested under a rupturing pressure of 3.5 atm (gauge). The diaphragms may be made from aluminium foil 0.1 and 0.15 mm thick, such as used in capacitors. In the seal at "*b*" a second vessel *5* is intended to collect the water which may be expelled from the seal back into the gas line, if the valve is not tight enough. Closed seals are designed for and are tested at a pressure of 22 or 32 atm (gauge), the maximum working pressure being 0.7 or 1.5 atm (gauge), respectively. This is done so as to ensure the necessary margin of safety, because an explosion of acetylene develops 13 times the initial pressure. Therefore:

$$p = (1 + 0.7) \times 13 - 1 = 22.1 \text{ atm (gauge)};$$
$$p = (1 + 1.5) \times 13 - 1 = 31.5 \text{ atm (gauge)}.$$

Table 8

Performance and Structural Dimensions of Closed Medium-pressure Water Seals

Characteristic	Seal type				
	station			manifold	
	I	II	III	I	II
Flow, m³/hr	Max. 0.8	Max. 1.5	Max. 3	Max. 10	Max. 35
Maximum pressure, atm (gauge)	1.5	1.5	1.5	1.5	1.5
Resistance at maximum flow rate, atm (gauge)	0.01	0.05	0.13	0 1	0.15
Gas velocity through valve, m/sec	1.8-2.5	8.3	16.5	13.9	15.8
Gas velocity through seal, m/sec	0.03-0.04	0.075	0.1	0.175	0.185
Weight of valve, grams	4	6	6	30	130
O. D. of valve ball, mm	—	15	15	25	40
Diameter of valve orifice, mm	10	8	8	16	28
I. D. of seal housing, mm	85	84	102	142	259
Wall thickness, mm . .	2	2.5	3	5	7
Diameter of hole under diaphragm, mm . . .	11	12	12	46	46
Water head above valve, mm	260	250	270	270	480
Total height of seal housing, mm	450	564	582	675	780
Water volume, litres . .	1.5	1.35	2.1	5.5	22
Gas space, litres	1.05	1.45	2.4	5.7	27
Diameter and number of holes in gas distributor, mm/holes . . .	—	—	—	—	3/72 in cap; 3/170 in splitter
O. D. and wall thickness of gas tube, mm . . .	—	21.25× ×2.75	21.25× ×2.75	42.25× ×3.25	42.25×3.25

The performance data and basic structural dimensions of commercially available medium-pressure water seals are summarised in Table 8.
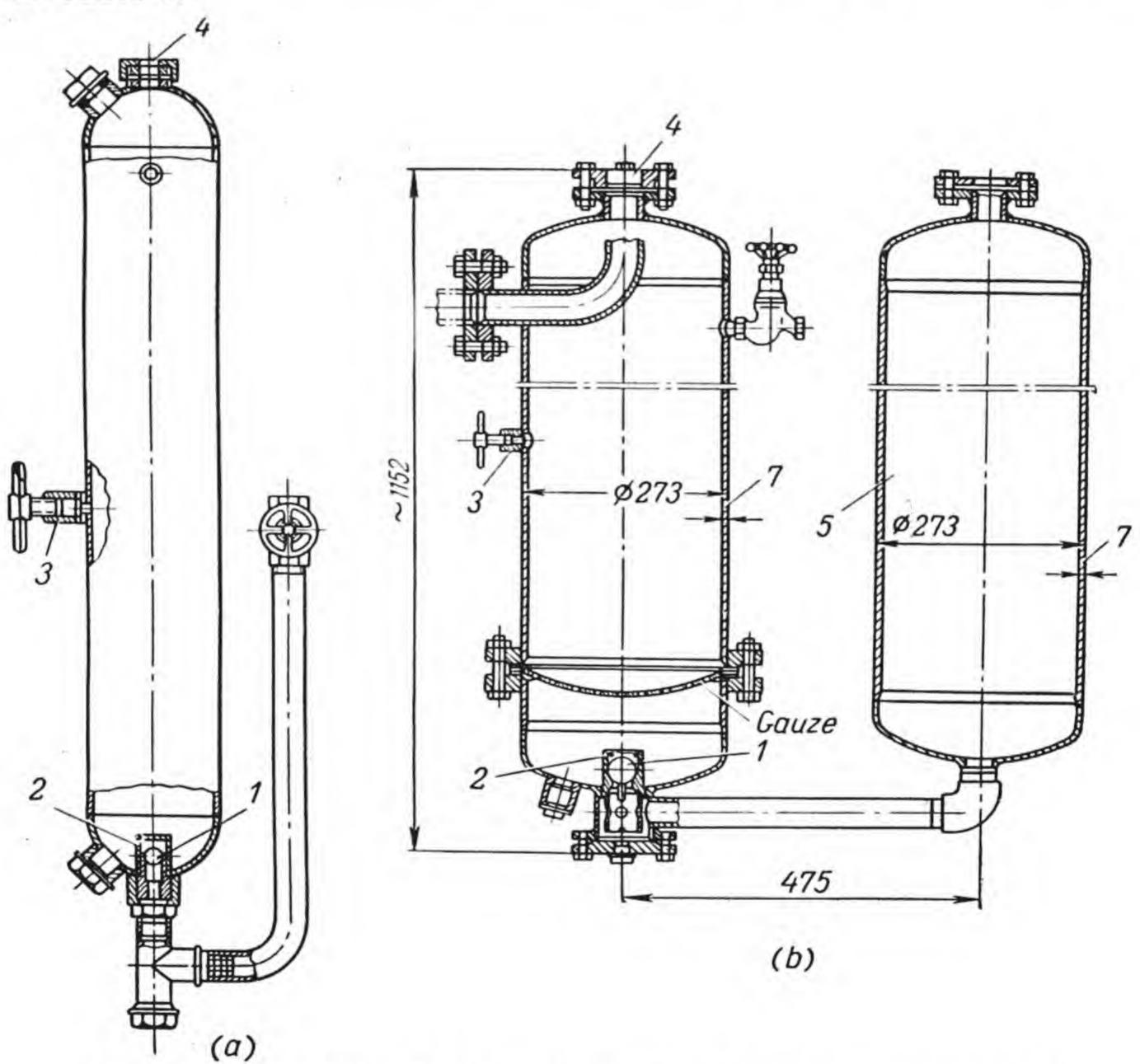
Fig. 23. Standard closed water seals for station and manifold service

8. Dry Safety Seals

In some cases, use is made of what is known as dry safety seals. One such seal, designed and tested by the Autogenous Welding Research Institute, consists (Fig. 24) of a steel body *1* into which is luted a porous cylinder *2* which is 25 mm· high and 52 mm in diameter, and made from a mixture of crushed grog and water glass. The cylinder is clamped in the body by a washer *3* which has holes near its periphery. On the ends of the body

are screwed caps *4* and *5* which have hose nipples placed on rubber gaskets. The cap *5* has a rubber ball nonreturn valve *6*, while the nipple in the cap *4* has a burst diaphragm *7*. The valve *6* prevents a reverse flow of oxygen into the acetylene line, while the porous cylinder damps the shock wave in the case of a flashback. The shock wave breaks the aluminium-foil diaphragm, and the explosive mixture is discharged into the atmosphere. Such a seal can pass up to 2 cu m of acetylene an hour at a pressure of up to 1.5 atm (gauge). Its resistance is equivalent to a head
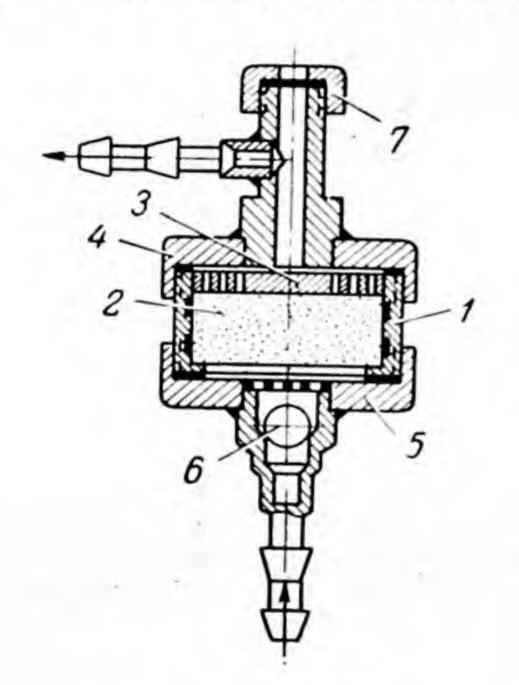
Fig. 24. Diagram of a dry safety seal

of 0.06 or 0.07 atm (gauge). Though this type of seal does not use any water, it can "freeze" at low ambient temperatures due to the condensation and freezing in the porous cylinder of the water vapours carried by the acetylene which comes from a generator.

9. Chemical Purifiers of Acetylene

The H_2S and PH_3 contained in acetylene are removed from it by materials in which the active ingredients are chromium or chlorine. The chromium or chlorine oxidise the volatile compounds of sulphur and phosphorus into nonvolatile substances — acids or salts.

The most common chemical acetylene purifier is known under the trade name of heratol. Heratol is a yellow-orange powder made from infusorial earth treated with an aqueous solution of $Na_2Cr_2O_7 \cdot 2H_2O$ and sulphuric acid. The infusorial earth and the solution are taken in a one to one proportion.

The composition of the charge and of heratol (in per cent by weight) is as follows:

	Charge	Heratol
$Na_2Cr_2O_7 \cdot 2H_2O$	17.1	11-13 (CrO_3)
Sulphuric acid (H_2SO_4)	22.9	17-20
Infusorial earth	50.0	45-55
Water	10.0	18-28
Total	100	100

CrO_3 is produced by the reaction of $Na_2Cr_2O_7 \cdot 2H_2O$ and sulphuric acid:

$$Na_2Cr_2O_7 \cdot 2H_2O + H_2SO_4 = Na_2SO_4 + 2CrO_3 + 3H_2O. \quad (III.7)$$

The chromium trioxide reacts with the impurities in the acetylene as follows:

(a) with H_2S:

$$8CrO_3 + 3H_2S + 9H_2SO_4 = 4Cr_2(SO_4)_3 + 12H_2O; \quad (III.8)$$

(b) with PH_3:

$$8CrO_3 + 3PH_3 + 9H_2SO_4 = 3Cr_2(SO_4)_3 +$$
$$+ 2CrPO_4 + H_3PO_4 + 12H_2O. \quad (III.9)$$

Used heratol is of greenish tint. It cannot be recovered for re-use, and has to be replaced. It takes 75 to 100 grams of heratol to purify one cubic metre of acetylene. Acetylene is purified better when dry than when moist.

Lumped purifiers have also been developed. Usually they are based on calcium hypochlorite whose chlorine is used as the active material.

10. Acetylene Generator Stations

An acetylene generator station is a suitable building, equipped with all the necessary plant and auxiliaries for the centralised generation and supply of acetylene to torches. Centralised acetylene supply is safe and economical and is therefore employed at all modern metal-working factories.

Acetylene generator stations can be classed into plants:

(a) generating free acetylene and manifolding it to where it is required over a pipeline at a pressure of up to 1.5 atm (gauge);

(b) producing dissolved acetylene;

(c) producing both free and dissolved acetylene;

(d) generating acetylene under a pressure of over 1.5 atm (gauge) for special applications (such as chemical processes).

By their gas-generating capacity acetylene stations can be classed into small (with an output of 25 m³/hr), medium (from 25 to 100 m³/hr) and big (over 100 m³/hr).

Apart from a generator building, an acetylene generating station includes a carbide storage room, a storage room for empty and filled cylinders and an acetone storage room. The main building accommodates the generator, the gas holder (unless it is part of the generator, or if the output of the generator is in excess of 25 cu m per hour), a service carbide storage room for not more than 2 tons of carbide, a room for opening carbide drums and services. A dissolved acetylene station will additionally include rooms for compressors, engines or motors, cylinder filling, cylinder repair and testing, and, where necessary, for filling the cylinders with acetone. These compartments or sections are separated from one another by partition walls of noncombustible materials, continuous from floor to roof or ceiling. Fig. 25 shows a typical acetylene generator station equipped with a Type ГРК-10 generator producing free acetylene.

Under the Fire Prevention Code in the Soviet Union, any acetylene generator station must be located at least 100 metres away from departments and equipment using an open light or fire, such as blast and open-hearth furnaces, forging shops, foundry shops, heat-treatment departments, and the like. Acetylene generator installations with an output of below 25 cu m per hour may be located in a lean-to attachment to basic production buildings or within these buildings. In such cases there must be no openings from the generator compartment into the main building, and the separation walls must be of fire-proof continuous construction.

Acetylene generator stations must be located at least 100 metres away from built-up areas and at least 50 metres away from separate houses. A generator station must have running water, electricity, heating, and railway sidings or motor roads laid to it. The heating must be by steam, hot water or other indirect system. Where an explosive mixture of acetylene and air may be formed, the wiring must be laid outside the generator building, and artificial lighting must be restricted to outside floodlights beaming through hermetically sealed windows. Sometimes use may be made of explosion-proof electrical equipment. The space about generators must be well ventilated by natural means. Big generator stations may have plenum ventilation using forcing fans. Sludge pits must be located outside of the generator station, covered by tightly fitting lids, and have escape pipes carried full size to above the crest of the roof.

B-B

Sludge pit

3500

3500

3600

3500

6500

2500

1

4

Storage room

5000

2

Generator room

Drum-opening room

B

B

8000

3000

3

Service carbide storage

Services

9000

4500
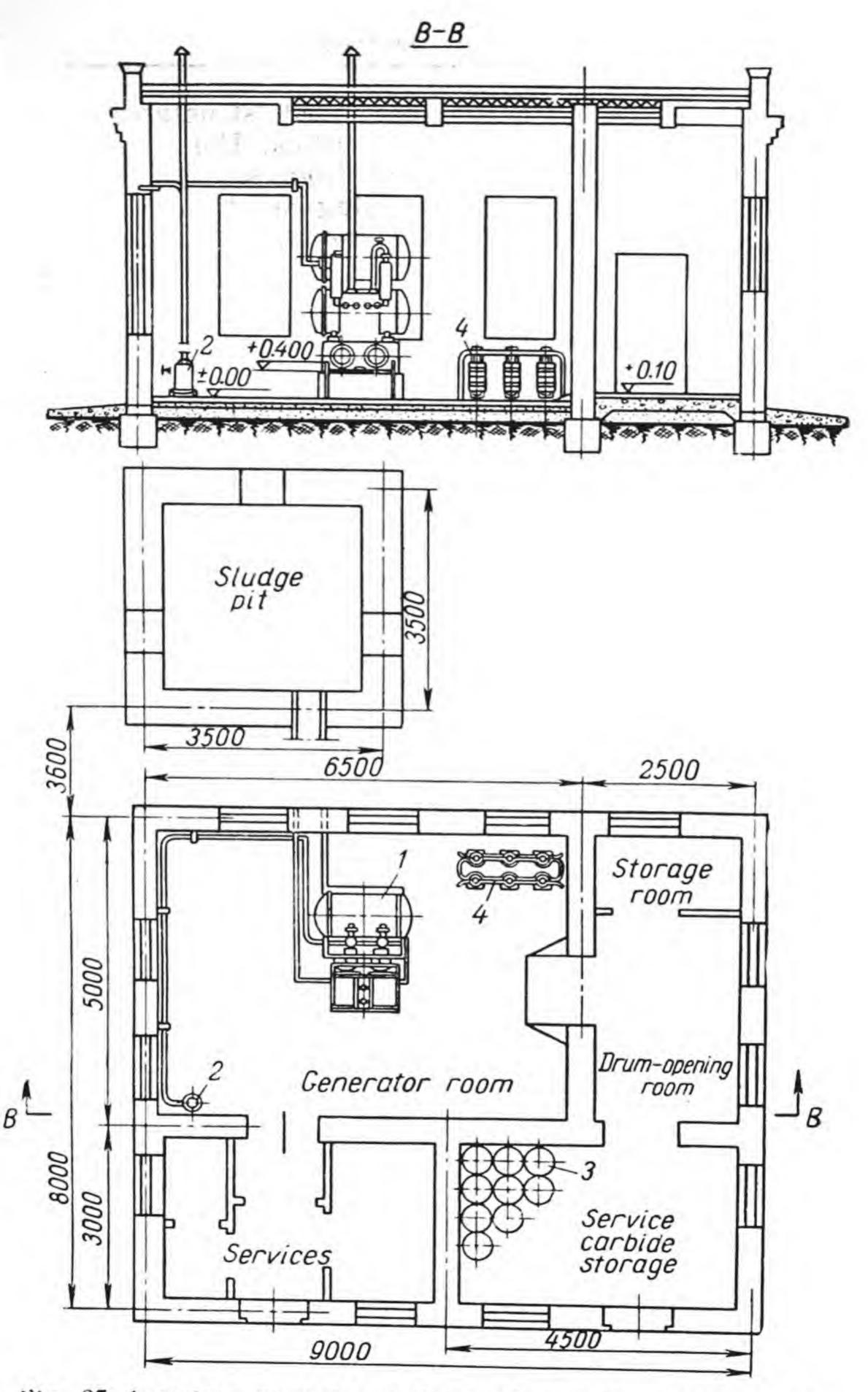
Fig. 25. Acetylene generator station producing free acetylene and incorporating a ГРК-10 generator of 10 m³/hr capacity:

1 — generator; 2 — drip pot; 3 — carbide drums; 4 — carbide tray racks

Generator houses are single-storeyed structures with light noncombustible ceilings and without attics. The exterior and interior walls and partitions are built from some fire-proof material, which may be brick, concrete, slag-concrete blocks, etc. The floor space of a generator building depends on the gas-generating capacity of the generators installed and must be not less than specified below:

Acetylene-generating capacity	Floor space
Max. 5 cu m/hr	8 sq m
6 to 10 cu m/hr . . .	16 sq m
11 to 20 cu m/hr . .	24 sq m
21 to 30 cu m/hr . . .	32 sq m
31 to 50 cu m/hr . . .	45 sq m
51 to 75 cu m/hr . . .	52 sq m
76 to 100 cu m/hr . . .	60 sq m
over 100 cu m/hr . . .	80 sq m

The floors must be asphalt-covered. In the cylinder filling, repair and testing rooms the floors may be from wood. Metal platforms, where used, must be covered with rubber mats to prevent sparking.

Each generator station must be within the protection zone of a lightning arrestor.

CHAPTER IV

FUEL GASES OTHER THAN ACETYLENE

1. General

Although acetylene is the most commonly used fuel gas, cheaper fuel gases may be used instead in many flame-processing applications. This above all is true of processes where the oxy-gas flame is employed to heat the metal to below the melting point of steel (flame cutting, flame hardening, flame heating for straightening, bending, etc.) and to weld low melting metals, and for torch soldering. The use of cheaper fuel gases instead of acetylene cuts down the cost of a flame metal-working process, simplifies the techniques and procedures and enhances safety to operating personnel.

The possibility and efficiency of using any fuel gas for flame metal-working processes depend primarily on the following of its properties and qualities:

(1) low heat value;
(2) specific weight;
(3) ignition temperature and flame velocity;
(4) flammable limits (per-cent gas in theoretical, optimum and working oxygen-gas mixtures);
(5) flame temperature;
(6) intensity of combustion and specific flame output;
(7) convenience and safety of production, transportation and utilisation.

This chapter deals only with some of these properties of fuel gases.

The low heat value, H_L, is the heat, in kcal, generated when 1 cubic metre or 1 kg of combustible is completely burned. For pure hydrocarbons and hydrogen it is a physical constant and is determined in gas calorimeters. For gas mixtures of known analysis the heat value in kcal/m³ (at 20° C and 760 mm Hg) can be determined by the equation:

$$H_L = 24H_2 + 80CH_4 + 206C_3H_8 + 140C_mH_n +$$
$$+ 28CO + 275C_4H_{10}. \qquad (IV.1)$$

In this equation, the constituents of the mixture are given in per cent by volume. The symbol C_mH_n designates the sum of the remaining high-molecular weight hydrocarbons in the gas. The coefficients preceding the constituents are obtained as $0.01\ H_L$ for each elementary combustible, with H_L expressed in kcal/m³.

Example: Coke-oven gas has the following composition: 59% H_2; 25% CH_4; 3% C_mH_n; 7% CO; 2% CO_2; 0.6% O_2; 3.4% N_2. Its low heat value is:

$$H_L = 24 \times 59 + 80 \times 25 + 140 \times 3 + 28 \times 7 = 4{,}029\ \text{kcal/m}^3.$$

The specific weight for mixtures of gases can be determined by the equation:

$$\gamma_m = (r_1\gamma_1 + r_2\gamma_2 + \cdots + r_n\gamma_n)0.01, \qquad\qquad (IV.2)$$

where $r_1, r_2, \ldots, r_n$ = content of the constituent, per cent by volume;

$\gamma_1, \gamma_2, \ldots, \gamma_n$ = specific weight of the constituent, kg/m³.

Example: Coke-oven gas has the following composition: 58.6% H_2; 25.2% CH_4; 2.4% C_3H_8; 7.1% CO; 2.4% CO_2; 0.6% O_2; 3.7% N_2.
The specific weights of the constituents (at 20° C and 760 mm Hg) are: H_2, 0.084; CH_4, 0.67; C_3H_8, 1.86; CO, 1.16; CO_2, 1.84; O_2, 1.33; N_2, 1.16 kg/m³.

By Eq. (IV. 2), the specific weight of the coke-oven gas in question will be $\gamma_m = (58.6 \times 0.084 + 25.2 \times 0.67 + 2.4 \times 1.86 + 7.1 \times 1.16 + 2.4 \times 1.84 + 0.6 \times 1.33 + 3.7 \times 1.16) \times 0.01 = 0.44$ kg/m³.

The theoretical ratio of oxygen (V_{ox}) to a combustible (V_c) necessary for complete combustion varies with the analysis of the combustible. For composite gases it may be determined by the relation:

$$\beta_{th} = \frac{V_{ox}}{V_c} = 0.01\left[0.5H_2 + 2CH_4 + 5C_3H_8 + \right.$$

$$\left. + \sum\left(m + \frac{n}{4}\right)C_mH_n + 0.5CO - O_2\right]. \qquad (IV.3)$$

Example: Shale gas has the following composition: 40% H_2; 14% CH_4; 4% C_3H_8; 20% CO; 15% CO_2; 0.5% O_2 and 6.5% N_2. The volume of oxygen necessary for the complete combustion of 1 cu m of combustible will be:

$$\beta_{th} = 0.01(10.5 \times 40 + 2 \times 14 + 5 \times 4 + 0.5 \times 20 - 0.5) = 0.775 \text{ cu m.}$$

2. Properties of Fuel Gases Other than Acetylene

The principal characteristics of, and uses for, acetylene substitutes in flame metal-working processes are summarised in Table 9. Their flammability (explosibility) limits in air-gas or oxygen-gas mixtures are given in Table 4 of Chapter II. A brief outline of these gases follows.

Hydrogen is normally a clear gas without colour or odour. The oxygen content in commercial hydrogen should not exceed 0.5 per cent (by volume). Commercially, hydrogen is produced by

(1) the electrolysis of water, (2) the steam-water gas process, (3) the thermal decomposition of methane or natural gas, and (4) the fractionation of coke-oven gas through refrigeration. In some cases use is made of portable hydrogen generators which produce hydrogen by the reaction of iron (filings) or zinc and sulphuric acid, or ferrosilicon, silicon and aluminium and an alkaline solution. Hydrogen can also be obtained by the reaction of H_2O and CaH_2 or a powdered mixture of ferrosilicon, caustic soda and lime.

In handling hydrogen, special care must be taken to keep all the equipment and gas lines as gas-tight as possible, as hydrogen is capable of finding its way through the tiniest cracks, forming explosive mixtures in the surrounding space.

Retort gas is a mixture of gases produced during the thermal decomposition of oil, oil products and residual oil in retorts at 720 to 740° C. The yield is 0.35 to 0.4 cu m per kilogram of fuel oil.

Retort gas has to be thoroughly purified as it contains sulphuric compounds which attack copper tips. An outfit for producing retort gas for flame metal-working processes has been successfully operated at the Urals Engineering Works for a number of years.

Refinery oil gas or *cracking still gas* is very near to retort gas in terms of properties. It is a by-product of oil pyrolysis and cracking at refineries. Under normal conditions both retort gas and refinery oil gas are without colour but have an obnoxious odour. The composition of refinery oil gas varies with the composition of the original oil and the conditions under which it was decomposed.

As the refinery oil gas is withdrawn from a cylinder, in which it is held partially liquefied, its composition varies, as the more volatile components vaporise first. To make the composition of the gas more constant and to prevent its condensation in pipelines, hoses and before torches at low ambient temperatures, use is sometimes made of an intermediate gas holder placed between the cylinder and the torch. Its capacity is usually 40 litres, i. e., that of a gas cylinder. The gas from the cylinder is delivered to the intermediate gas holder through a pressure regulator at a pressure of 3 or 4 atm (gauge) and from the intermediate gas holder to the torch through another pressure regulator. Instead of an intermediate gas holder this purpose can be served by a pipeline of the requisite capacity, which should be not less than 40 litres per cylinder.

Natural gas is a mixture of hydrocarbons issuing from natural gas or oil wells, consisting mainly of methane (92-99 per cent) which is normally a gas without odour or colour.

6*

Gas and its composition (In per cent by volume)	Sp. weight at 20° C and 760 mm Hg, kg/m³	Low heat value at 20° C and 760 mm Hg, kcal/m³	Flame temperature in oxygen, °C
Acetylene (C₂H₂) . . .	1.09	12,600	3,150
Hydrogen (H₂)	0.084	2,400	2,100
Retort gas *. 28-36% H₂; 42-53% CH₄; 17-19% CₘHₙ; 5-12% CO . .	0.65-0.85	7,500-8,000	2,300
Refinery oil gas *, 12% H₂; 28% C₂H₄ and C₃H₆; 50% C₂H₆, C₃H₈ and CH₄; 10% CO₂, CO, etc.	0.63-1.45	9,800-13,500	2,300
Natural gas, 97.8% CH₄; 0.9% C₃H₈ and C₂H₆; 1.3% N₂ and CO₂	0.7-0.9	7,500-7,900	2,000
Coke-oven gas *, 50-59% H₂; 25-30% CH₄; 1.8-3% C₂H₄ and other unsaturates; 5-7% CO; 6-13% N₂ and CO₂; 0.5-0.8% O₂	0.4-0.55	3,500-4,200	2,000

Table 9

Fuel Gases

Oxygen-to-gas ratio		Delivery to torch	Application
for complete combustion (theoretical)	In torch		
2.5	1-1.3	Over hoses and pipelines from generators or cylinders	Welding, cutting and heating of all metals
0.5	0.3-0.4	In cylinders at 150 atm (g), over pipeline, or from portable generator	Cutting, brazing; welding of lead and aluminium; hardening
Eq. (IV. 2)	1.2-1.5	Over pipeline from retort	Cutting, brazing; welding of cast iron and steel up to 3 mm; brass and aluminium; hardening
Ditto	1.5-1.6 (2 in cutter)	In cylinders at 150 atm (g) or over pipeline	Ditto
Ditto	1.0-1.5	In cylinders at 150 atm (g), or over pipeline	Cutting, brazing; welding of lead, aluminium; hardening
Ditto	0.75-0.8	Over pipeline	Ditto

Gas and its composition (in per cent by volume)	Sp. weight at 20° C and 760 mm Hg, kg/m³	Low heat value at 20° C and 760 mm Hg, kcal/m³	Flame temperature in oxygen °C
City gas *, 6-10% H_2; 60-70% CH_4; 5-7% C_2H_4; 4-6% CO; balance, N_2, CO_2, etc.	0.84-1.05	4,100-5,000	2,000
Shale gas *, 25-40% H_2; 14-17% CH_4; 10-20% CO; 10-20% CO_2; 4-5% C_2H_6, etc.; 22-25% N_2; up to 1% O_2	0.74-0.93	3,000-3,400	2,000
Propane-butane mixture, 85% C_3H_8; 12% C_4H_{10}; up to 3% C_2H_6	1.92	21,200	2,100
Petrol (liquid)	0.695-0.760 kg/lit	10,200-10,600 kcal/kg	2,500-2,600
Kerosene (liquid) . . .	0.816-0.841 kg/lit	10,000-10,200 kcal/kg	2,450-2,500

* The composition of gas mixtures is only approximate and may vary within broad limits. Their

Coke-oven or *by-product gas* is a mixture of gases produced during coke production from coal. It is widely used as a fuel at iron and steel works and as a source material for many chemicals.

Shale gas is produced by the gasification of combustible shales in special generators in the presence of air. The composition of shale gas varies with the source material and the process used.

Propane and *butane* are fractions of natural or refining-still gases. They have high critical temperatures and low critical pressures and can therefore be liquefied at relatively low pressures and positive temperature.

Commercially, a mixture of propane and butane is usually employed. This mixture is transported in welded steel cylinders of 23 and 33 kg capacity, in tank trucks of 3 tons capacity, and tank cars of 50 tons capacity of liquid gas.

Pure propane can be drawn directly from the vapour phase

Table 9 (continued from page 85)

Oxygen-to-gas ratio		Delivery to torch	Application
for complete combustion (theoretical)	In torch		
Ditto	1.2-1.3	Ditto	Ditto
Ditto	0.7	Ditto	Ditto
Ditto	3-3.5	In cylinders at up to 15 atm (g)	Ditto; plus welding of steel up to 2 mm and cast iron
2.6 m³/kg	1.1-1.4 m³/kg	From service tank at 2 or 3 atm (g)	Steel cutting; hardening; brazing
2.55 m³/kg	1.7-2.4 m³/kg	Ditto	Ditto

specific weights and heat values may also vary within broad limits.

of the cylinder at an ambient temperature of down to —25° C, and the propane-butane mixture at down to +8° C. At the lower ambient temperature, the gas has to be drawn from the liquid phase. For this purpose the cylinder should be turned with its valve down or to fit the valve with a siphon tube. When drawing the liquid phase from the cylinder, it is necessary to have an evaporator for liquid gases.

Petrol and *kerosene* are used in flame metal-working processes in vaporised form. Torches burning them are fitted with suitable evaporators heated either with the preheating flame or electrically.

Before use, kerosene should preferably be thawed and filtered through felt and a column of caustic soda (NaOH) to remove water and solid resin (residual tar) particles, naphthenic and sulphonaphthenic acids and their salts which, on burning, leave ash and solid residues which plug the evaporator in the torch.

CHAPTER V

COMPRESSED GAS CYLINDERS, VALVES AND PRESSURE REGULATORS

1. Cylinder Classification and Construction

Compressed, liquefied and dissolved gases at higher than atmospheric pressure are transported and stored in steel cylinders varying in capacity. In flame metal-working processes, widest use is made of cylinders with a capacity of 40 litres. The seamless cylinder types manufactured of carbon and alloy steel to the relevant U.S.S.R. Standard are presented in Table 10.

Table 10

Type desig-nation	Pressure, kg/cm²			U. T. S., kg/mm²	Yield point, kg/mm²	Elon-gation, %	Impact strength, kg-m/cm²
	Design, P_d	Test					
		hydrau-lic	pneuma-tic	minimum			
Carbon steel cylinders							
100	100	50% above working pressure	working pressure	65	38	15	—
150	150	225	150				
200	200	300	200				
Alloy steel cylinders							
150Л	150	225	150	90	70	10	10
200Л	200	300	200				

Type 150 and 150Л cylinders may be used for oxygen, hydrogen, nitrogen, methane, compressed air and rare gases. Compressed air and methane may also be stored and transported in Type 200 and 200Л cylinders. Carbon dioxide is usually compressed in Type 150 cylinders, while Type 100 cylinders are used for acety-

lene, ammonia and other gases compressed to as high as 100 atm (gauge).

40-litre cylinders have the following dimensions:

O. D., mm	219
Wall thickness, mm: .	
Type 100 and 150Л .	5.2
Type 150 and 200Л .	7
Type 200	9.3
Length (Types 150 and 200Л), mm . . .	1,390
Weight, kg	60

The U.S.S.R. Pressure Vessel Code stipulates that cylinders shall be painted blue for oxygen, white for acetylene, black with a white stripe for commercial argon, white on the upper half and black on the lower half for pure argon, black for air, dark-green for hydrogen, and red for other fuel gases.

(a) *Seamless cylinders* are fabricated from seamless steel hollows rolled in several passes on special mills or drawn (cupped) from flat blanks on special presses. To make a cylinder, a blank is cut from a seamless hollow, and then stamped to form the throat and the bottom of the cylinder to the desired shape. The cylinder throat has a threaded conical opening which receives the shank of a cylinder valve.

The valve is covered by a protector cap screwed into a steel ring on the cylinder throat. The protector cap protects the valve from mechanical injury in transit and handling. The cylinder throat is burred to retain the threaded ring in place. For stability in an upright position, a square shoe is shrink-fitted onto the bottom of each cylinder.

As distinct from other cylinders, those intended for acetylene are filled with a porous mass and acetone and must therefore meet special requirements of a relevant standard. Acetylene cylinders are usually the Type 100. The filler (activated birch-tree charcoal with a bulk weight of 225 g/lit) should be so compacted in the cylinder that its tensity is 280-310 grams per litre of cylinder capacity, and soaked with 225 or 230 grams of acetone per litre of cylinder capacity.

(b) *Welded cylinders* may be manufactured by any welding process. Before they may be allowed to fabricate cylinders, welders have to pass qualification tests in accordance with the U.S.S.R. Pressure Vessel Code. The quality of welded joints is determined by testing specimens cut from test plates welded

concurrently with the cylinders and by means of X-ray and gamma-ray inspection.

Welded cylinders are employed for liquid propane and butane, but may also be used for acetylene. They are fabricated from steel Cт. 3 with an ultimate tensile strength of 34-45 kg/mm² and an elongation of not less than 22 per cent.
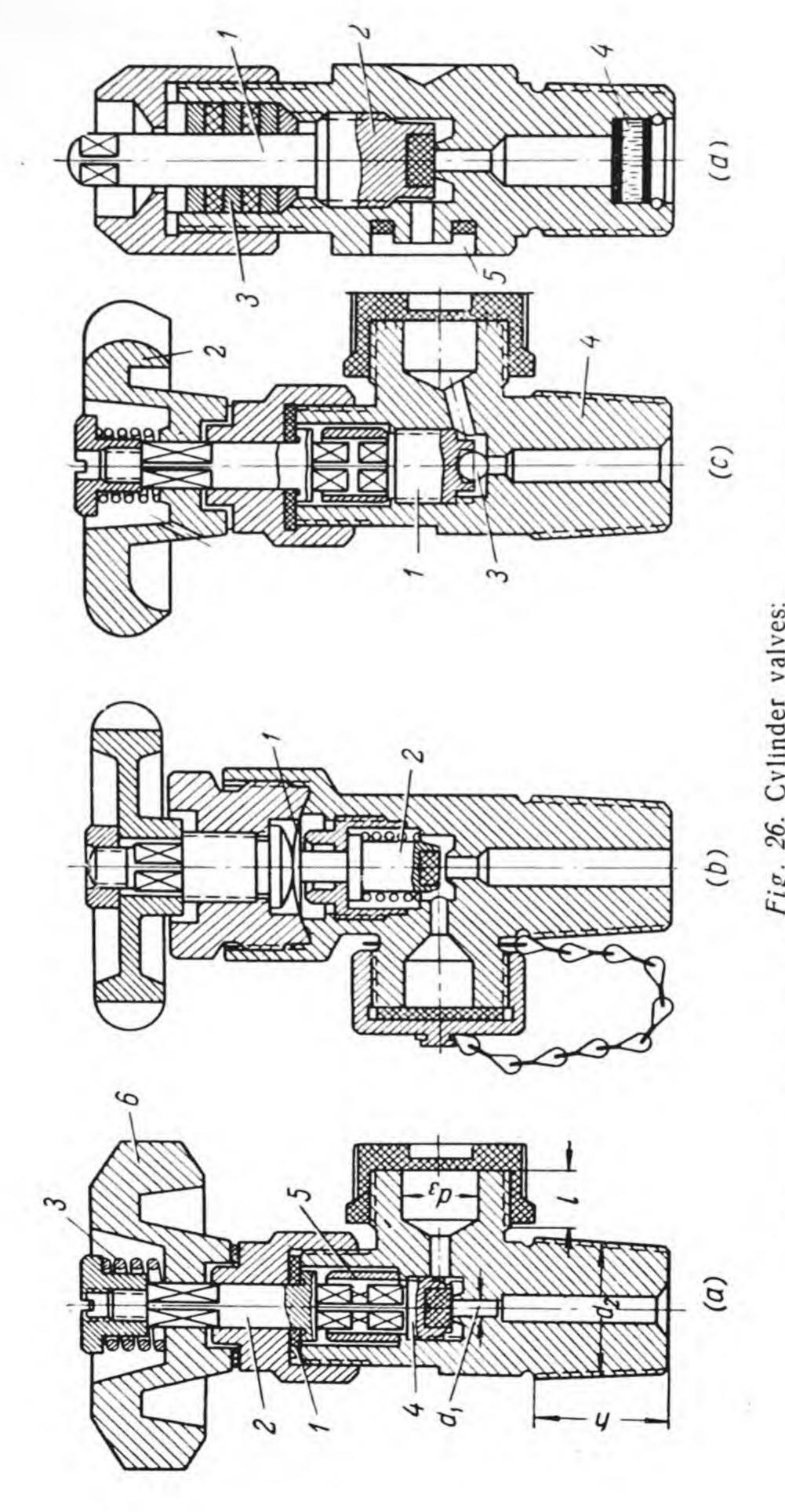
2. Cylinder Valves

An oxygen valve (Fig. 26*a*) has a seal in the form of a fibre ring *1* held in place by a spindle *2* which bears the pressure of a spring *3* and of the oxygen (when the valve plug *4* is opened). Rotation of the spindle is transmitted to the threaded valve plug *4* by a bushing *5* with a square through hole. All valve parts coming in contact with oxygen are made from brass and are degreased during assembly with dichlorethane, trichlorethane or carbon tetrachloride to prevent corrosion and burning with oxygen. The handwheel *6* may be made of steel, cast iron, aluminium alloy or plastic material. The seal must be made from top-quality fibre. Where the shoulder of the spindle bears against the fibre ring, it must be thoroughly polished. If not, the fibre ring will be abraded and may be ignited with compressed oxygen due to friction or the adiabatic heat when the valve is opened suddenly, with the result that the internal valve parts may be burned or melted out. To reduce friction, the fibre ring must be immersed in molten paraffin and held there at 70° C for 40 minutes, and excess paraffin must be removed afterwards.

In a diaphragm valve (Fig. 26*b*) the seal is perfectly tight. The diaphragm *1* is made from a sheet (0.1 or 0.15 mm thick) of phosphorous bronze or stainless steel. Drawbacks of this type of valve are that the gas will completely escape from the cylinder should the diaphragm burst, and that the service life of the diaphragm is short where the valve has to be opened and closed very often.

In a ball-type valve (Fig. 26*c*), the valve plug can be easily closed tight by rotating the handwheel *2* without any spanner which is indispensable when using the valve shown in Fig. 26*a*. The ball *3* is made from ЭИ-229 stainless steel. The threaded valve plug is made from aluminium-manganese bronze БрАМц-9-2, which fact reduces friction and binding in the body *4* of the valve, which is made from brass ЛС-59-1.

Acetylene valves (Fig. 26*d*) are made from steel and have a spindle *1* turned integral with the valve plug *2*. The gland *3* is packed with raw pigskin impregnated with solid grease. The valve

Fig. 26. Cylinder valves:

a — oxygen valve; *b* — diaphragm-type oxygen valve; *c* — ball-type oxygen valve; *d* — acetylene valve

has a filter *4* and no side connection. Instead, it has a recess *5* to which a pressure regulator or the tube of a filling manifold can be connected by a suitable collar.

3. Cylinder Testing

According to the U.S.S.R. Pressure Vessel Code, cylinders must be inspected and tested at filling plants every five years. Cylinders for aggressive gases as well as transportation cylinders for motor vehicles burning compressed gases must be tested every two years. Filling plants obtain permission to test cylinders from local Pressure Vessel Safety authorities which register the stamps of the respective filling plants. A periodic test includes external and internal inspection, a check on the weight and capacity of the cylinder, and a hydraustatic test. Before inspection, the cylinder must be emptied of all gas. The purpose of the inspection is to determine the condition of the cylinder walls, to reveal traces of corrosion, cracks, scabs, dents and other injuries.

The capacity of a cylinder is determined as the difference between its weight when filled with water at 20° C and when empty. If the loss in weight is anywhere from 7.5 to 10 per cent and the increase in capacity is from 1.5 to 2 per cent, the cylinder must be rated for a pressure 15 per cent lower than previously and stamped accordingly. If the loss in weight is more than 20 per cent and the increase in capacity is more than 3 per cent, the cylinder must be discarded. During a hydraustatic test, a cylinder is subjected to a pressure which is 1.5 times its working pressure for one minute, after which the pressure is reduced to the working one, and the cylinder is inspected.

Cylinders for dissolved acetylene, filled with a porous mass, are only given a pneumatic test with 97-per cent nitrogen at a pressure of 30 atm (gauge). During this test, the cylinder must be submerged in a water bath to a depth of at least 1 metre. In addition, the porous filler must be inspected (through the cylinder throat) every year so as to determine whether or not it has settled down.

The certificate data and periodic test data are stamped on the dome of each cylinder.

4. Corrosion of Oxygen Cylinders

When dry, oxygen oxidises iron but slowly. In an oxygen cylinder only a very thin layer on the inside of the cylinder shell is affected, as the oxides thus formed produce a continuous film which prevents further rust formation. In the case of damp oxygen,

moisture may accumulate in the cylinder, thus promoting corrosion.

Rust formation may be approximately described as follows:

$$
\begin{array}{cc}
\mid\!\rightarrow\,\rightarrow O_2 & \mid\!\rightarrow\,\rightarrow O_2 \\
Fe + H \cdot OH' \rightarrow Fe\,(OH) + H \cdot OH' \rightarrow \\
\leftarrow \quad\quad \leftarrow\!\mid \quad\quad \leftarrow \quad\quad \leftarrow\!\mid
\end{array}
$$

$$
\begin{array}{c}
\mid\!\rightarrow\,\rightarrow O_2 \\
\rightarrow Fe\,(OH)_2 + H \cdot OH' \rightarrow Fe\,(OH)_3. \\
\leftarrow \quad\quad \leftarrow\!\mid
\end{array}
$$

Iron hydroxides are a loose and porous mass pervious to oxygen, and the latter can easily reach deep into the wall of the cylinder, thus affecting it throughout its thickness. The presence in the moisture of ions of chlorine and sodium (sodium chloride) also promotes rust formation. Flaws on the inside of the cylinder shell, such as scabs and overlaps, may be attacked to a point where the cylinder wall may be completely destroyed.

As tests at a number of filling plants have shown, when the water with which oxygen cylinder shells are lubricated is free from common salt, no noticeable corrosion can be detected on cylinders even after 20 years' service. For better safety, however, it is essential to inspect cylinders on the inside thoroughly for all possible sources of rust formation and to discard defective ones.

5. Explosion of Cylinders

The potential energy of compressed gas released in the explosion of a cylinder can be determined by the equation of the work performed by the adiabatic expansion of gases, known from thermodynamics:

$$
L = \frac{p_1 V_1}{k-1}\left[1 - \left(\frac{p_2}{p_1}\right)^{\frac{k-1}{k}}\right], \tag{V.1}
$$

where L = energy released by a gas expanding from p_1 to p_2, in kg-m;

p_1 = pressure of the gas in a cylinder, in kg/m²;
p_2 = ambient pressure, in kg/m²;
V_1 = volume of the cylinder, in cu m:
$k = \dfrac{c_p}{c_v}$ = adiabatic coefficient.

For example, when V_1 = 40 litres = 0.04 cu m, p_1 = 150 × × 10⁴ kg/m², p_2 = 1 × 10⁴ kg/m², we obtain

$$
L = \frac{150 \times 10^4 \times 0.04}{1.4 - 1}\left[1 - \left(\frac{1 \times 10^4}{150 \times 10^4}\right)^{\frac{1.4-1}{1.4}}\right] = 154{,}000 \text{ kg-m.}
$$

When a cylinder explodes, this energy is released within a split second and can therefore inflict heavy damage.

Sometimes, though seldom, compressed gas cylinders do explode. An inquiry into the causes of an explosion is difficult as there is usually lack of knowledge of the conditions under which the explosion was set off. Nevertheless, in the light of the known explosions, it seems safe to think that the most common causes have been inherently defective cylinders (hidden cracks, scabs, overlaps, laminations, blowholes, etc.) as well as internal stresses and structural defects in the cylinder metal.

As the temperature drops to —60° C, the impact strength of carbon steels sharply diminishes from 10 or 11 to 0.5 kg-m/cm². An increased carbon content in a steel aggravates its brittleness at low temperatures still more. This calls for utmost care in handling filled cylinders at low ambient temperature. The presence of coarse-grained structures or locked-up stresses in the cylinder metal increases the possibility of a cylinder explosion due to rough handling at low temperatures.

Another cause of an explosion may be the formation of an explosive oxygen-gas mixture in the cylinder. This may take place when oxygen is compressed into a cylinder which has been filled with a fuel gas or when a fuel gas finds its way into an oxygen cylinder. An explosion may further be set off by oil getting into an oxygen cylinder, as oil self-ignites in compressed oxygen.

6. Classification of Pressure Regulators

Pressure regulators, or more properly pressure reducing valves, are attached to the cylinder valves of oxygen, acetylene and other cylinders or a pipeline system to reduce the pressure to a suitable working pressure at the torch. They also maintain a nearly constant pressure at the torch irrespective of its variations before the torch.

A pressure regulator consists essentially of a nozzle, a seat, a flexible diaphragm, and springs which control the lift of the seat depending on the pressure in the regulator chamber.

Diagrammatically, a pressure regulator is shown in Fig. 27. The high-pressure gas is admitted into the high-pressure chamber *4* through a nipple *1*, whence it is passed into the low-pressure chamber *8* fitted with a diaphragm *11*.

As it passes through the nozzle, the high-pressure gas loses some of its pressure in overcoming the resistance of the nozzle.

The nozzle is closed by the seat *5* held down by an opposing (smaller) spring *3*, and forced away by a large or controlling spring *9* which acts on the seat through a pin *12*. The pressure of

the spring *9* on the diaphragm *11* and, consequently, the travel of the seat *5* can be adjusted by means of the screw *10*. When the flow of gas through the regulator is reduced, the pressure in the low-pressure chamber *8* increases, and the diaphragm is forced down, overcoming the spring *9* so that the seat *5* reduces the inflow of gas into the chamber *8* until the pressure in the chamber is decreased, and a balance is struck between the forces acting upon the diaphragm *11* and the seat *5*. As more gas is discharged
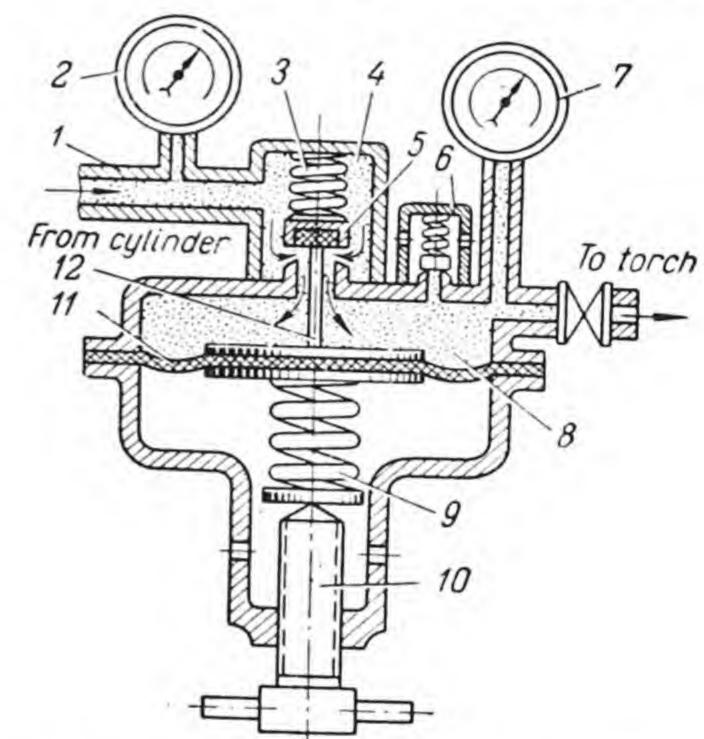
Fig. 27. Diagram of a reverse-acting pressure regulator

from the regulator into the torch line, the pressure in the chamber *8* is reduced, the spring *9* forces the seat *5* further away, and the inflow of gas through the nozzle is increased. If more gas is admitted into the chamber *8* than is withdrawn from the regulator, the pressure in the chamber *8* rises again; the spring *9* is compressed, and the seat *5* is lowered by the opposing spring *3*.

The pressure regulator has a blow-off valve *6* which protects the diaphragm against breakage, should the pressure in the low-pressure chamber drop too much, a high-pressure gauge *2*, and a low-pressure gauge *7*.

The pressure regulators employed in flame welding and cutting can be classed into several groups:

   (a) the cylinder, station and manifold types;
   (b) the direct-acting and reverse-acting types;
   (c) the nonlever and lever types;
   (d) single-stage and double-stage regulators;
   (e) oxygen, air, acetylene, hydrogen, propane and other regu-
lators;

(f) high-pressure (up to 150 atm, gauge) and medium-pressure (15 to 30 atm, gauge) regulators.

Cylinder regulators are usually designed to pass 40 to 60 cu m of oxygen per hour at a working pressure of 15 atm (gauge) or up to 5 cu m of acetylene per hour at a pressure of up to 1.5 atm (gauge).
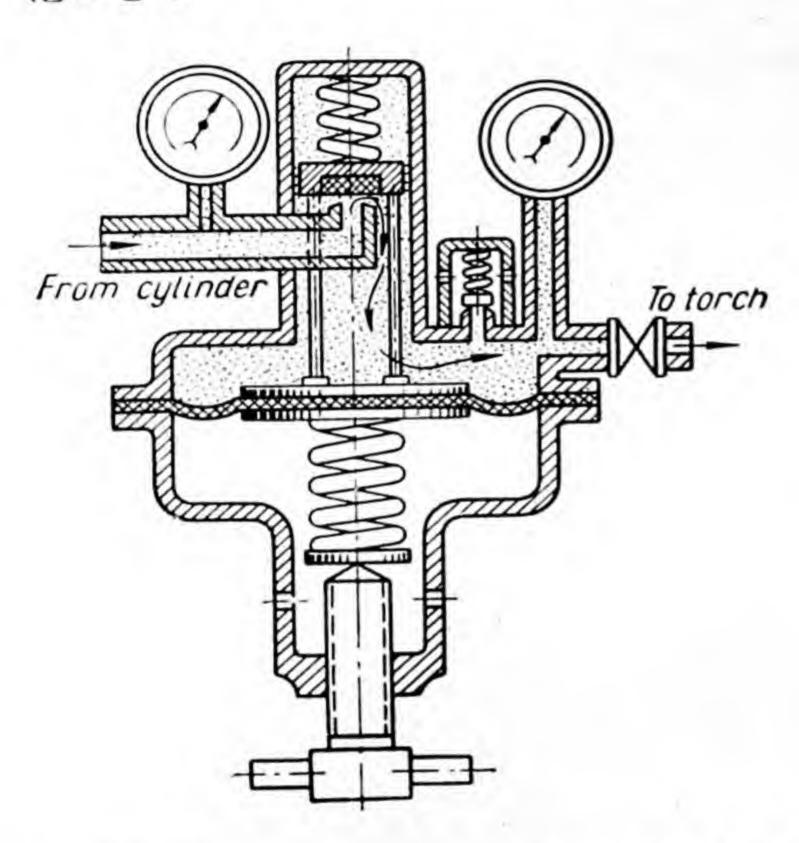
Fig. 28. Diagram of a direct-acting pressure regulator

Station pressure regulators are mounted on shop oxygen lines and operate at an inlet pressure of 15 to 30 atm (gauge). They are designed for the maximum consumption of oxygen by a given welding station, which averages 30 to 40 cu m per hour.

Manifold regulators can pass up to 250 cu m per hour at a pressure of up to 25 atm (gauge). In some cases, such as in hogging machines, use is made of regulators capable of handling as much as 1,500 cu m or more oxygen an hour.

A reverse-acting regulator (Fig. 27) is one in which the seat closes with the pressure of the incoming high-pressure gas. In a direct-acting regulator the seat closes against, rather than with the pressure (Fig. 28).

The balance of forces in a pressure regulator is described as follows:

for a direct-acting regulator:

$$p_2 = \frac{N_n - \Delta_n h + f_{noz} p_1}{F_e} \qquad (V.2)$$

for a reverse-acting regulator:

$$p_2 = \frac{N_n - \Delta_n h - f_{noz} p_1}{F_e}. \qquad (V.3)$$

Notation:

p_2 = absolute pressure in the low-pressure chamber, kg/cm^2;

$N_n = N_c - N_o - N_d$ = resultant force of the controlling spring (N_c), opposing spring (N_o) and diaphragm (N_d), kg;

$\Delta_h = \Delta_c + \Delta_o + \Delta_d$ = aggregate stiffness of the controlling spring (Δ_c), opposing spring (Δ_o) and diaphragm (Δ_d), kg/mm;

h = lift of the seat, mm;

f_{noz} = cross-sectional area of the nozzle;

p_1 = absolute pressure before the regulator, kg/cm^2;

F_e = effective area of the diaphragm, sq cm:

$$F_e = 0.26 (D^2 + DD_1 + D_1^2), \qquad (V.4)$$

where D = outside diameter of the diaphragm;
D_1 = diameter of the central stiffener of the diaphragm.

From the equation of force balance it follows that when p_1 varies from 150 to zero, p_2 in direct-acting regulators is reduced and in reverse-acting regulators is increased. In other words, a direct-acting regulator has a drooping pressure characteristic and a reverse-acting regulator has a rising pressure characteristic. Variations in $\Delta_n h$ may be neglected, as this quantity is insignificant in comparison with N_n and $f_{noz} p_1$.

Regulators with one stage of regulation are called single-stage and those with two stages of regulation, double-stage. Double-stage regulators ensure a more constant working pressure, as all variations in the high pressure are absorbed in the first stage and do not affect the pressure in the second stage. In addition, two-stage regulators are more substantial and are less liable to over-cooling and "freezing" at the higher rates of gas delivery. The disadvantages of two-stage regulators are more complicated design, a higher limit of pressure reduction (see below), and a greater weight of metal.

Regulators for different gases differ in their pressure ratings in accordance with the limits of pressure for the respective cylinders. Their inlet nuts and nipples also differ so that these regulators may not be attached to the wrong cylinders. In addition, regulators are painted different colours for the same purpose.

7. Performance of Pressure Regulators

The *outlet pressure* and *capacity* of a regulator should be chosen to suit a given application. The capacity of a pressure regulator is governed by the cross-sectional area of the nozzle, the difference between the pressure before and after the seat, and the area of the outlet nozzle. As a rule, the volume rate of flow through the seat nozzle may be several times the one through the outlet nozzle under a given pressure in the low-pressure chamber. Therefore, in defining the capacity of a regulator it is customary to specify the diameter of the nozzle through which the gas is directed to the torch.
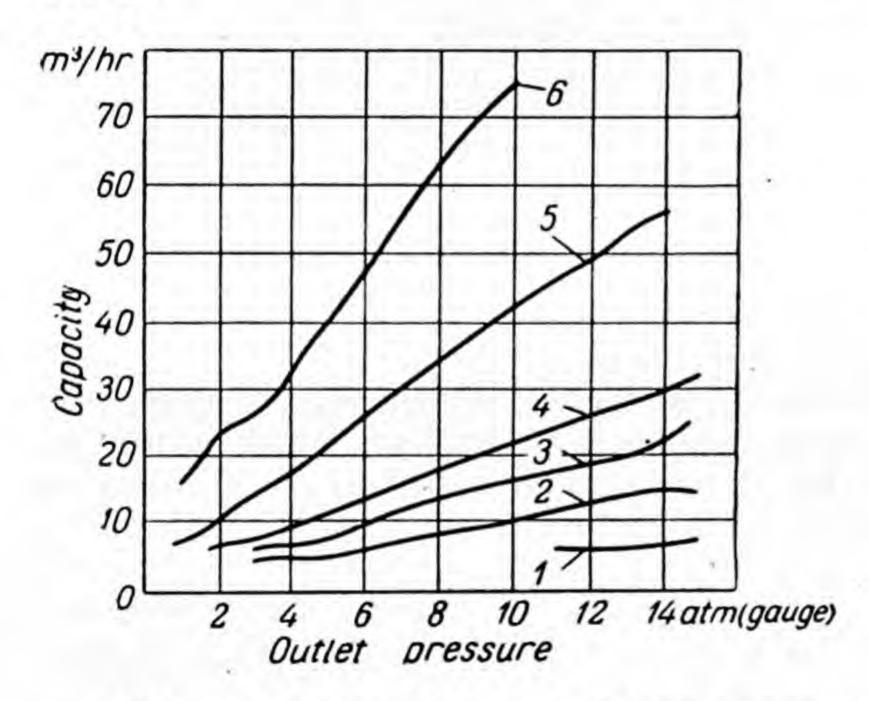

Fig. 29. Gas flow through a pressure regulator plotted against the pressure in the low-pressure chamber and the outlet nozzle diameter. The diameter of the outlet nozzle is as follows:
1 — 1 mm;   *2* — 1.4 mm;
3 — 1.8 mm;   *4* — 2 mm;
5 — 2.8 mm; *6* — 4 mm

The capacity of a regulator increases with the diameter of the outlet nozzle and the pressure in the low-pressure chamber (Fig. 29). As the pressure in the cylinder is reduced, the volume rate of flow through the seat nozzle is increased. Therefore, for the weight rate of flow through the regulator to be maintained constant, the cross-sectional area of the nozzle should be increased. This is accomplished automatically by lifting the seat (Fig. 30). When all the gas is withdrawn from the cylinder the pressure in it sharply drops, and the lift of the seat is increased in proportion. This corresponds to the lower limit of reduction (see below).

The regulation sensitivity is the variation in the pressure of the low-pressure chamber when the adjusting screw of the controlling spring is given a quarter of a turn. The regulation sensitivity is governed by the ratio of the effective area of the diaphragm to the cross-sectional area of the nozzle, the thread pitch of the adjusting screw, and the stiffness of the controlling spring. For station

regulators, the regulation sensitivity ranges between 0.5 and 1.5 kg/cm² for each quarter of a turn of the adjusting screw.

The total operating differential may be defined as the amount by which the outlet pressure of a regulator must increase when no gas is withdrawn from it before the seat shuts off the supply of the high-pressure gas.

Let the outlet pressure at a given discharge rate be p_2 and the outlet pressure when no gas is discharged be p_2'. Then their difference will be $\Delta p_2 = p_2' - p_2$. The quantity Δp_2 in called the *total operating differential* and is expressed either in atmospheres or in per cent of the initial value of p_2.

$$\Delta p_2 = \frac{p_2' - p_2}{p_2} \, 100. \qquad (V.5)$$

The total operating differential gives a qualitative measure of regulator design and workmanship. The total differential will be
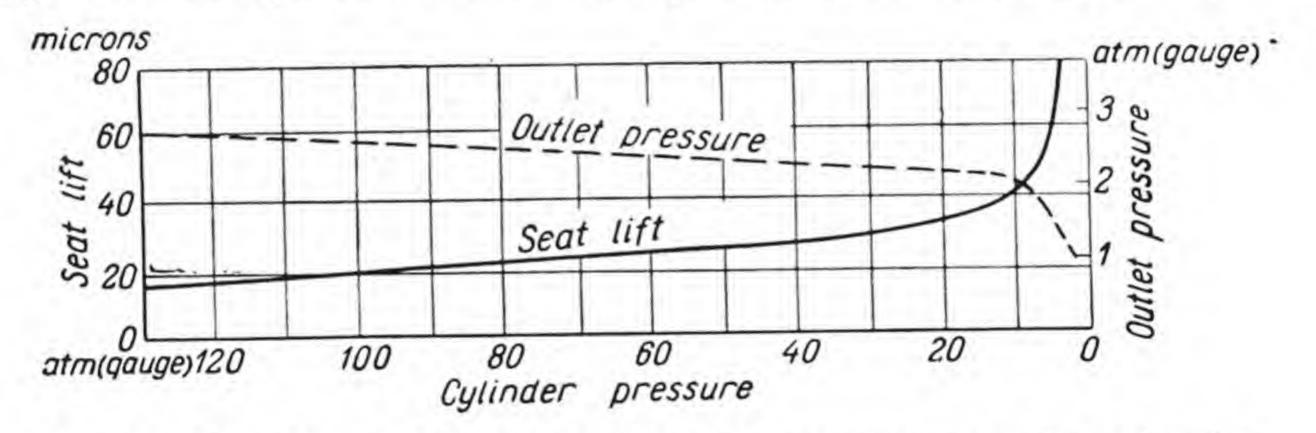
Fig. 30. Seat lift and outlet pressure in a pressure regulator related to to cylinder pressure

smaller when the seat is machined better, there is less friction in the regulator, its parts have a lower inertia, and the diaphragm is more elastic. The total operating differential is also affected by the outlet pressure, the rate of gas discharge, and the ratio of the effective area of the diaphragm to the cross-sectional area of the seat. In a reverse-action regulator the percentage differential increases with increasing pressure in the low-pressure chamber and decreases with increasing ratio of the effective area of the diaphragm to the seat area. In existing station regulators the total differential ranges between 15 and 30 per cent. A greater total differential may result in a dangerous rise of the pressure in the low-pressure chamber and the burst of the diaphragm, should the blow-off valve fail to operate for one reason or another.

In reverse-acting regulators, the total differential is lower than in direct-acting regulators because the high-pressure gas coming

7*

from the cylinder speeds up the closure of the seat when no gas is withdrawn from the regulator.

The reduction limit indicates the lowest pressure in the cylinder or pipeline at which the outlet pressure begins to decrease rapidly. When the reduction limit has been reached, no further lift of the seat can compensate for the decrease in the gas quantity entering the low-pressure chamber, and the pressure in it drops quickly. The minimum gas pressure before the regulator, p_{1min}, at which the reduction limit is reached, is 2 or 2.5 times the outlet pressure. The reduction limit corresponds to the critical pressure ratio, $p_2/p_{1min} = 0.528$.

8. Commercial Regulator Designs

In the Soviet Union, pressure regulators are available in a variety of designs for the various applications in flame metal-working processes. Fig. 31 shows a Type PK-50 (*a*) and a Type PK-53 (*b*) oxygen regulator. Both regulators are of single-stage, reverse-acting design. The high-pressure gas from the cylinder enters the regulator by a nipple *1*. The seat *2* closes over the nozzle *3* screwed into the regulator body *4*. The seat is moved by a pin *5* bearing upon a diaphragm *6*. The lift of the seat can be adjusted at will by means of an adjusting screw *7* and a controlling spring *8* located under the regulator cover *9*. The PK-53 regulator, which has now superseded the PK-50, has an additional nozzle screwed into the inlet nipple to prevent the seat from freezing when the oxygen from the cylinder carries moisture with it.

РД-2А acetylene regulators and PB-55 hydrogen regulators are designed along the same lines. The РД-2А regulator differs from an oxygen regulator in the size of the controlling and opposing springs, the diameter of the seat, and in that it has a clip instead of a union nut. The union nut on the hydrogen regulator has an L. H. pipe thread of 21.8 mm diameter, 14 turns per inch. Fig. 31*c* shows a Type РД-2А acetylene regulator.

Fig. 32 presents a Type KPP-50 two-stage manifold regulator in which the second stage uses the pressure of the gas entering the chamber above the diaphragm instead of a controlling spring. The high-pressure oxygen enters the regulator by the nipple *1*, passes through the first-stage seat *2*, flows up the passage *3*, passes through the second-stage seat *4*, and is delivered into the line through the outlet nipple *5* under the working pressure. From one side, the diaphragm *6* in the second stage is acted upon by the working-pressure gas, and from the other side by the gas supplied to the chamber between the diaphragm *6* and the cover *7* through the passage *3* and a valve (omitted in Fig. 32). The
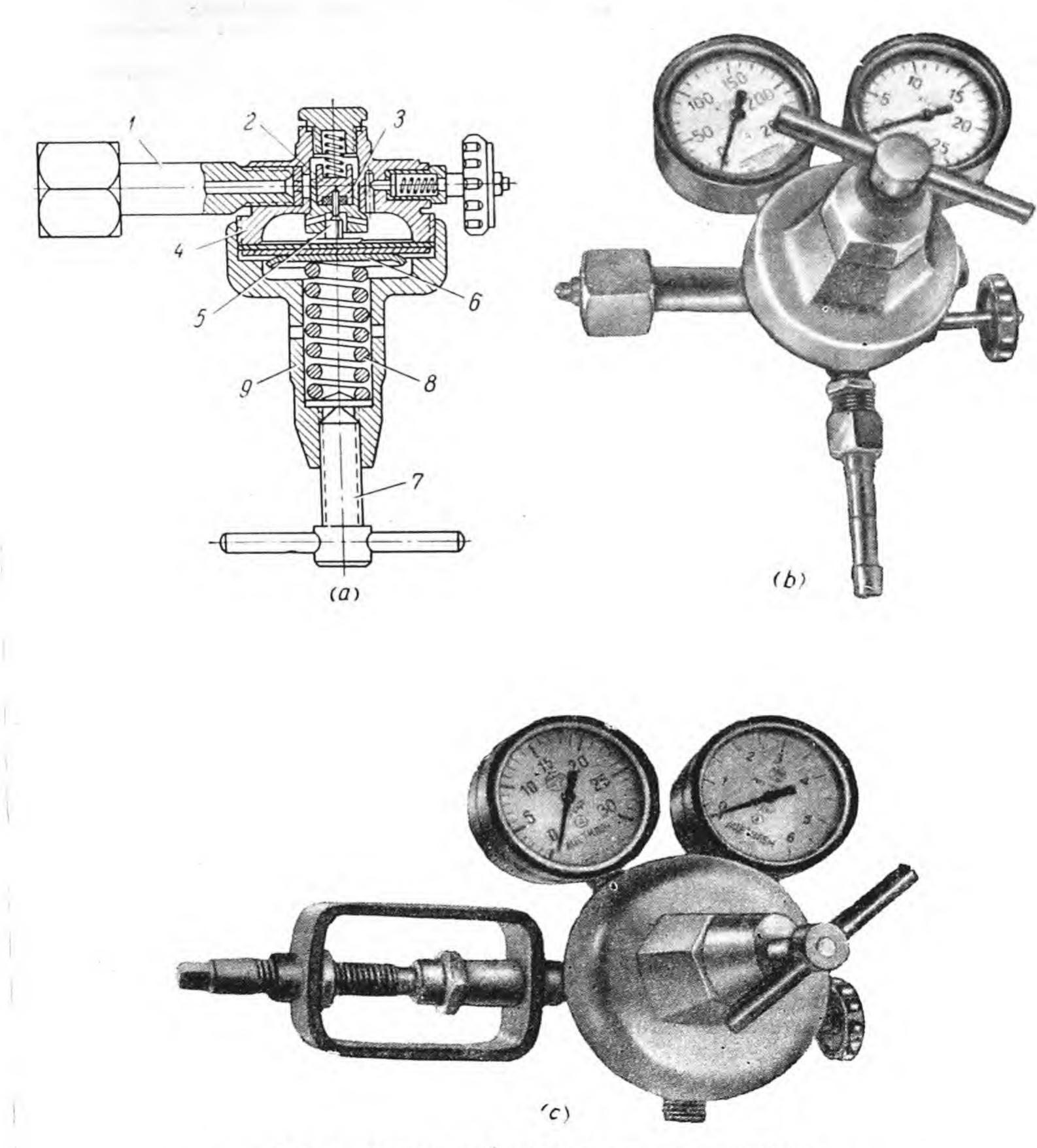
Fig. 31. Oxygen and acetylene pressure regulators:
a — Type PK-50; *b* — Type PK-53; *c* — Type РД-2A acetylene regulator

pressure from this chamber is released by the valve *8*. The pressure in the second stage can be adjusted at will by increasing or reducing the pressure in the chamber above the diaphragm by means of the two valves.
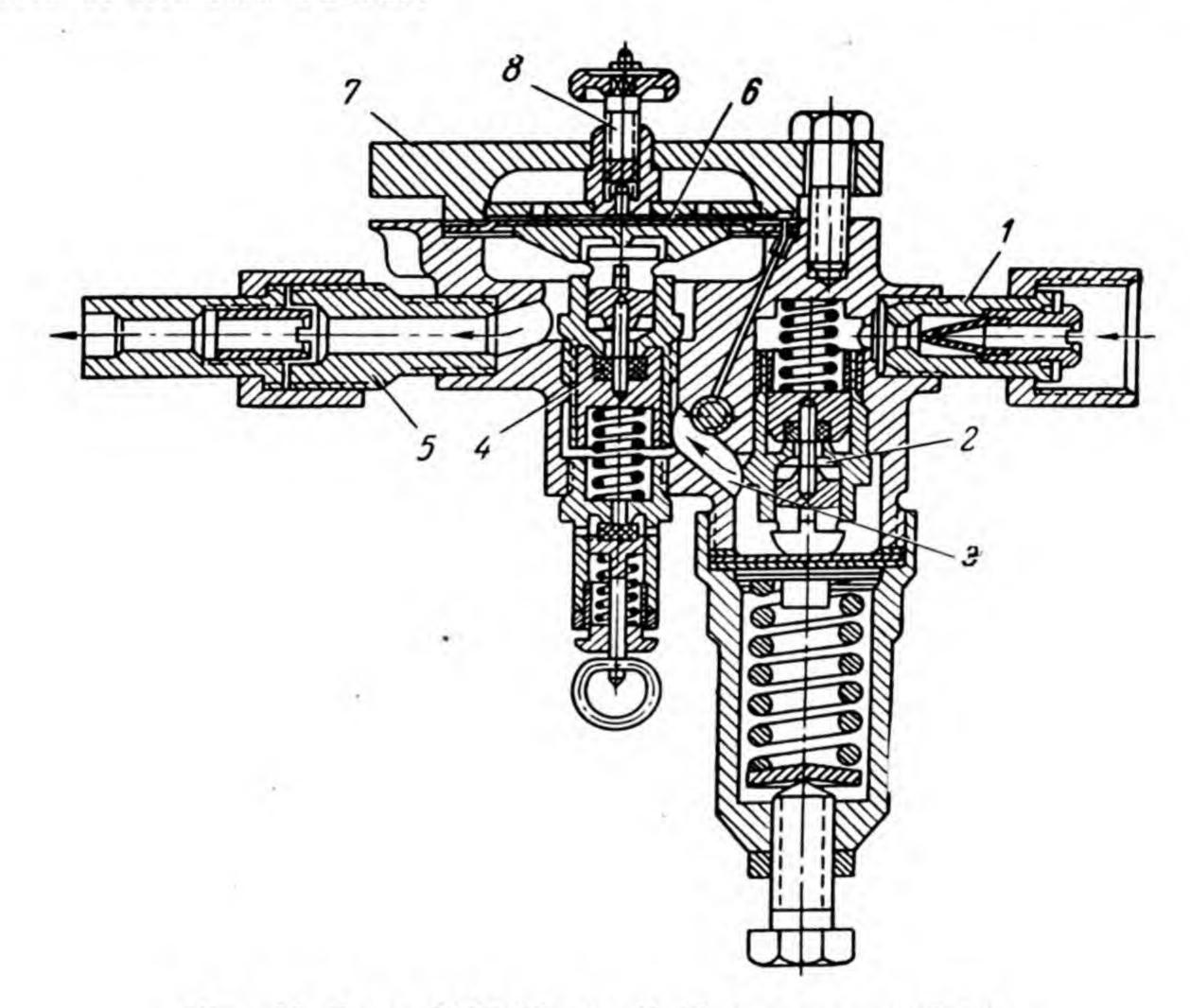
Fig. 32. Type KPP-50 manifold oxygen regulator

9. Safety Rules for Regulator Operation

(1) Burning of Oxygen Regulators. Cases are known when the internal parts of an oxygen regulator burned after the shut-off valve of a cylinder or manifold had been opened suddenly. Parts from organic materials (rubber, hard rubber and fibre, etc.) usually burn first. Their burning in the presence of compressed oxygen causes the metal parts, such as springs, pressure disks, cast-iron caps, etc., to burn too. The brass parts in the zone of burning are melted, and the regulator is completely destroyed.

When a shut-off valve is opened suddenly, the gas before the seat in the regulator is subjected to practically adiabatic compression from 1 to 150 atm (gauge), which may bring about a considerable rise in the temperature of the gas. Theoretically, the

temperature of a gas subjected to adiabatic compression will be

$$T_2 = T_1 \left(\frac{p_2}{p_1}\right)^{\frac{k-1}{k}} = 293 \left(\frac{151}{1}\right)^{\frac{1.4-1}{1.4}} = 1,258 \,^\circ K \text{ or } 985 \,^\circ C.$$

This is high enough for hard rubber to ignite. True, the actual temperature will be lower owing to heat losses. However, hard rubber can ignite at 400° C. A hard-rubber seat will catch fire more eagerly if it has a rough, fibrous or porous surface where it comes in contact with the nozzle shoulder.
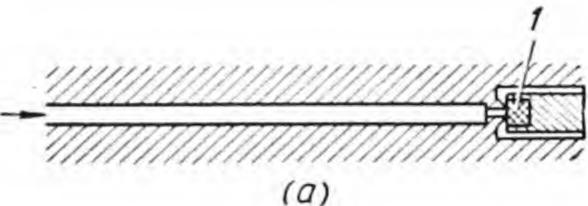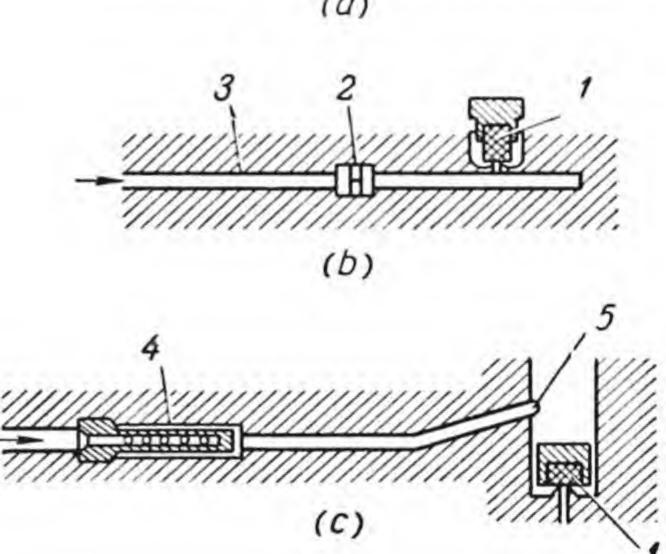
Fig. 33. Arrangement of oxygen passages in a regulator:

1 — hard-rubber seat; *2* — perforated heat-guard; *3* — elongated passage for heat abstraction through walls; *4* — heat-absorbing perforated throttle; *5* — expansion chamber.

An investigation has shown that regulator design has a direct bearing on the possibility of a jet of compressed and heated oxygen igniting hard-rubber parts. In particular, this may occur if the whole of the compression heat generated when an oxygen valve is opened suddenly is allowed to reach the hard-rubber parts of the regulator (Fig. 33a). This possibility can be reduced by placing a heat-guard (which may be a copper disk with fine holes) in the way of the oxygen flow or extending the passage before the seat (Fig. 33b). In either case, some of the compression heat will be abstracted through the regulator body. A still better approach is the arrangement shown in Fig. 33c, where the jet of oxygen is allowed to expand in a chamber above the seat before reaching it.

Another cause of ignition in a pressure regulator may be the electrostatic charge formed in the jet of oxygen flowing through the regulator nozzle at supersonic speed. The formation of an electrostatic charge may be promoted by metal particles, droplets

of moisture or specks of dust present in the stream of oxygen. The potential difference between the particles and the orifice may be as high as 6,000 or 7,000 volts. Experiments have shown that a potential difference of 2,000 or 2,500 volts is sufficient to set off a spark discharge through oxygen.

Last but not least, the parts of an oxygen regulator may be burned owing to the fact that some oil or grease has been allowed to come in contact with oxygen inside the regulator after repair, inspection, etc.

(2) "Freezing" of Oxygen Regulators. The throttling of the oxygen flow cools the regulator, and the water vapour contained in the oxygen may form ice crystals which will clog the orifices. This is most likely to occur at the higher rates of oxygen discharge, low ambient temperatures, high inlet pressure and small regulator size.

The freezing of regulators can be prevented by making them bigger, by using two stages of oxygen expansion, and by preheating and desiccating the oxygen before it enters a regulator. A variety of oxygen desiccators have been developed, which are installed between cylinder valve and regulator and use chemical substances, such as blue vitriol, Cu_2SO_4.

A frozen regulator may be warmed either by taking it into a heated room or with hot water or steam. Neither an open flame nor red-hot metal objects should be used for the purpose. After warming up, the regulator must be blown out to remove such moisture as may be left in it.

(3) Leaky Regulator Seats (Pressure Creep). Sometimes the high-pressure oxygen continues to enter the low-pressure chamber, although the nozzle is fully closed and the diaphragm spring is fully released. This may be due to the fact that the seat does not cover over the nozzle tightly. The causes of this condition may be the damaged surface of either of them, misalignment of the seat in its carriage, or foreign matter on the contact surface.

As a result, the pressure in the low-pressure chamber increases so that the diaphragm and the Bourdon tube of the low-pressure gauge may burst, especially if the outlet valve happens to be closed or the blow-off valve fails to operate in time. Every measure must therefore be taken to prevent or eliminate the cause of leakage through the seat and nozzle at once.

CHAPTER VI

GAS PIPELINES AND LINE ACCESSORIES

1. Oxygen Pipelines

For pressures up to 15 atm (gauge) welded steel pipes may be used for oxygen systems while for pressures between 16 and 30 atm (gauge) seamless steel pipes are recommended. Seamless steel pipes may be used to convey oxygen at over 30 atm (gauge) only when the pipeline is laid and buried in the ground. Otherwise, seamless copper pipes should be employed.

The reason why copper pipes should be used is that damp oxygen at elevated pressure easily oxidises steel pipes. The electrostatic discharge likely to occur in a stream of oxygen, as well as the ignition of scale or rust particles due to friction against the pipe walls, may also cause the steel to burn.

An electrostatic discharge is most likely to take place immediately past the pipe valve, for the oxygen flows through it at the critical speed when the valve is being closed or opened. In lines working at below 30 atm (gauge) it will be well advised to replace the steel pipe immediately past the valve by a length of red-copper or brass piping.

All valves and fittings used on oxygen lines must be made from bronze or brass. Fittings from stainless steel will stand corrosion, but may ignite, as all grades of stainless steel can burn in oxygen, although not so vigorously as low-carbon structural steel. Joints in steel pipelines for oxygen must be welded, and in copper lines welded or brazed. Flanged and threaded joints may only be resorted to where absolutely necessary (say, where valves or other fittings are installed on a line).

When a line is being laid and prior to welding its joints, each length of pipe must be degreased on the inside by repeatedly pushing through it a swab soaked with a solvent. The entire line must again be degreased before it is put into service. After degreasing surplus solvent must be blowed out with hot air, nitrogen or steam. The solvent may be pure carbon tetrachloride. Neither alcohol nor petrol should ever be used, as their traces may form

an explosive mixture with oxygen and lead to an explosion in the line. Dichlorethane vapours are likewise capable of forming an explosive mixture with oxygen. Vapours of carbon tetrachloride and dichlorethane are poisonous, and care must be exercised in handling them.

2. Acetylene Pipelines

Acetylene piping is classed into three groups: low-pressure (up to 0.1 atm, gauge), medium-pressure (from 0.1 to 1.5 atm, gauge) and high-pressure (over 1.5 atm, gauge). Welding shops use the first two groups of acetylene piping. High-pressure piping is employed at dissolved acetylene stations.

Acetylene lines are laid with steel seamless pipes jointed by welding. Flanged joints may be used only where acetylene generators, line fittings and manifolds are connected. Threaded joints should be avoided. It will be a sound policy to make high-pressure acetylene lines from stainless steel, as rust may set off an explosion of acetylene.

Acetylene piping must be laid away from furnaces, hearths or equipment with hot surfaces. Nor should it be laid in a common trench or duct with oxygen piping or electrical cables. When acetylene and oxygen piping is laid in a common trench, they must be separated by a continuous wall. When laid in the trench or duct with inert or fuel gas pipelines, acetylene piping must have supports of its own and be spaced at least 250 mm away from other piping.

In welding shops or departments, acetylene piping may be run in parallel with the oxygen line, using a common wall or common pillars, but separate hangers or brackets. Acetylene lines must be painted white, and caution signs must be stenciled on the walls of the respective trench or duct. It is essential that every acetylene line be solidly earthed. The ducts accommodating acetylene piping must be equipped with escape or relief pipes. As a measure against a detonation wave originating and propagating within an acetylene line, the pipe diameter must not exceed 50 mm for medium pressures and 15 mm for high pressures. Where acetylene has to be delivered at a high volume rate of flow, several lines must be laid instead of a single one of a large diameter.

High-pressure acetylene lines must be equipped with burst diaphragms installed every 25 metres and at the ends of the line. The diaphragms must be tested for a rupturing pressure of about 1.5 times the maximum working pressure of a given line. It is a good plan to set up escape pipes above the diaphragms and

other safety devices and to keep constant watch on their condition.

Acetylene piping is usually laid with a slop so that moisture may run off into drip pots installed at the lowest points in the line.

Before putting it into service, every acetylene line for a working pressure of 1.5 atm (gauge) should be tested hydrostatically under a test pressure $p_{test} = 13 (p_{work} + 1) - 1$, and those for over 1.5 atm (gauge), under a test pressure twice its working pressure, but not lower than 50 atm (gauge). After moisture has been blown out from an acetylene line, it should be tested for tightness with air at one and a half times its maximum working pressure but not lower than 1 atm (gauge). Before acetylene may be admitted into a line, the latter must further be blown out with nitrogen or carbon dioxide. A line may be considered clean if the oxygen content in the blown-out gas does not exceed 3.5 per cent. The blown-out gas must be discharged into the atmosphere so as not to contaminate the air in the shop.

3. Pipeline Calculation

Pipelines are calculated to cater for the highest volume rate of flow of a given gas and the permissible pressure drop. The volume rate of flow varies with the job on hand and the number of the welding (or cutting) stations supplied by a given line.

The pressure drop in a straight circular pipeline can be determined by the equation:

$$\Delta h = Kl \frac{V_0^2}{d^5} \frac{\rho}{p_{av}}, \qquad (VI.1)$$

where Δh = head loss, mm (water gauge);

    l = length of the pipeline, m;

    d = I. D. of the pipeline, mm;

    V_0 = flow through the pipeline at 20° C and 760 mm Hg, m³/hr;

    ρ = density of acetylene compared with air;

    p_{av} = average gas pressure in the pipeline, atm (abs) equal to

$$p_{av} = \frac{736}{760} \left(\frac{p_{in} + p_f}{2} \right) + 1,$$

where p_{in} and p_f are the initial and final pressures of the gas in kg/cm²;

K is coefficient of friction. For diameters from 10 to 100 mm, $K = 2.5 \times 10^5$. For acetylene, the density as compared with air is 0.905. Therefore:

$$\Delta h = 2.5 \times 10^5 l \frac{V_0^2}{d^5} \frac{0.905}{p_{av}} = 2.25 \times 10^5 \frac{V_0^2}{d^5} \frac{l}{p_{av}},$$

and

$$d = \sqrt[5]{\frac{2 \cdot 25 \times 10^5 V_0^2 l}{\Delta h p_{av}}} = 11.8 \sqrt[5]{\frac{V_0^2 l}{\Delta h p_{av}}}. \qquad (VI.2)$$

The average velocity of the gas in the pipeline (m/sec) will be:

$$w = \frac{V_0 4 \times 10^6}{3,600\pi \, d^2 p_{av}} = 354 \frac{V_0}{d^2 p_{av}}. \qquad (VI.3)$$

With d found by Eq. (VI.3), we have a check on the calculated diameter of the pipeline in terms of the average velocity of the gas:

$$d = \sqrt{\frac{354 V_0}{w p_{av}}} = 18.8 \sqrt{\frac{V_0}{w p_{av}}}. \qquad (VI.4)$$

For low- and medium-pressure acetylene lines the average velocity of the gas should lie within the following limits: for service lines intended for pressures up to 0.1 atm (gauge) $w = 3$ to 4 m/sec; for pressures from 0.1 to 1.5 atm (gauge), $w = 4$ to 8 m/sec; for main lines for pressures up to 0.1 atm (gauge), $w = 1$ to 2 m/sec; and for pressures from 0.1 to 1.5 atm (gauge), $w = 2$ to 4 m/sec.

In determining the head loss in a pipeline, it is usual to express friction due to valves and fittings in terms of additional length of straight pipe, adding this to the actual pipe length to get the equivalent pipe length. The total pressure loss is usually 5 to 10 per cent of the initial pressure.

Pipeline calculation can be greatly simplified by the use of the chart shown in Fig. 34 in which $A = \frac{\Delta h p_{av}}{l}$ and $B = w p_{av}$.

In the calculation of high-pressure piping use is made of the equation

$$d = \sqrt[5]{\frac{12 \cdot 5 \beta G^2 l \times 10^{-4} RT}{\gamma \, \Delta p \, p_{av}}}, \qquad (VI.5)$$

where $d =$ I. D. of the pipeline, mm;
  $G =$ hourly rate of gas flow, kg/hr;
  $l =$ length of the pipeline, m;
  $\gamma =$ specific weight of the gas, in kg/m³, at T in °K and at p_{av} in atm (abs) in the pipeline;

$\backsim p =$ permissible pressure loss in the pipeline, kg/cm², equal to the difference between the initial and final pressure across a given length, or $\Delta p = p_{in} - p_f = (0.002$ to $0.05)$ p_{in};

$\beta = \dfrac{2.86}{G^{0.144}}$ is the coefficient of friction.
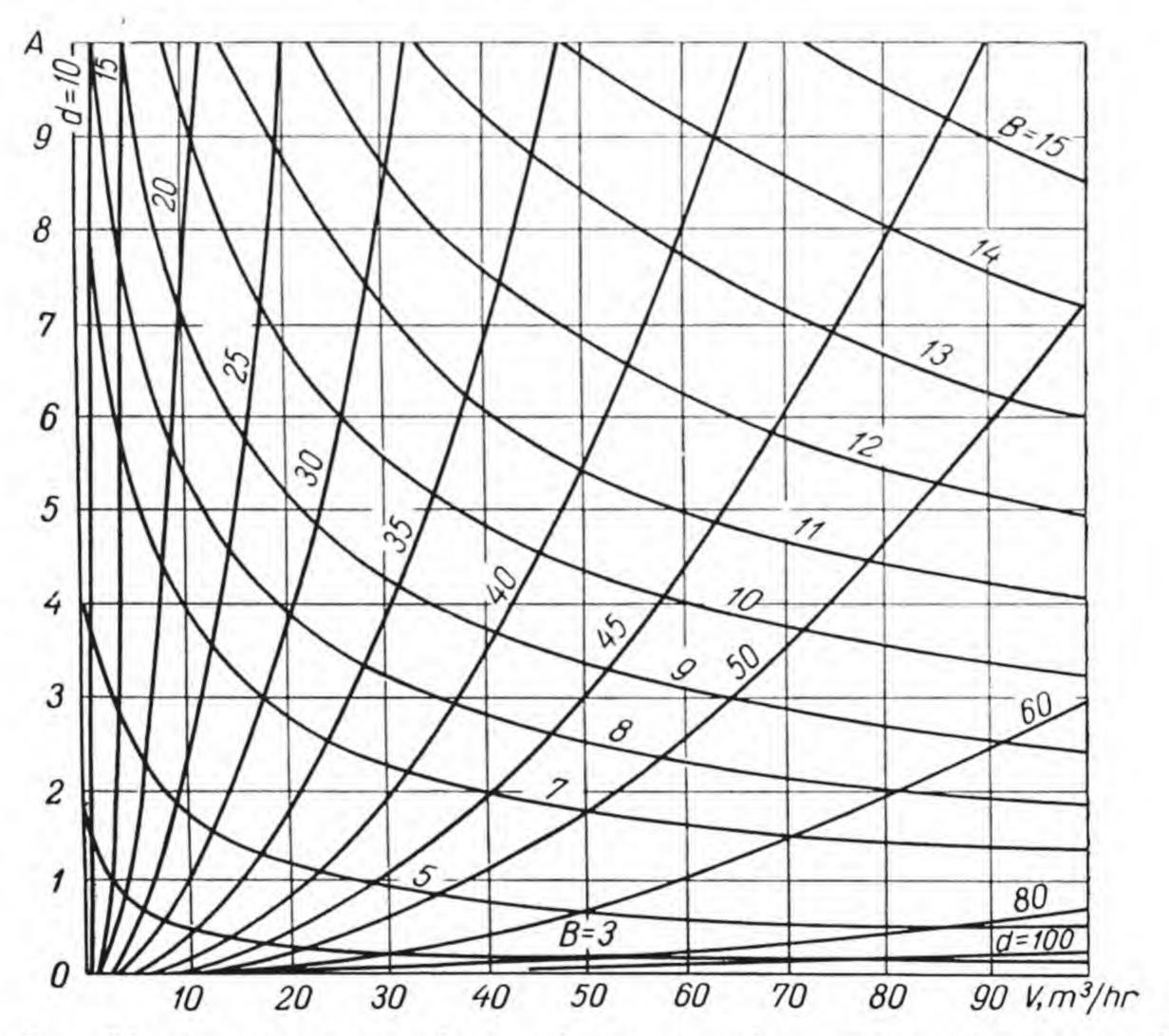
Fig. 34. An alignment chart relating acetylene piping diameter and volume rate of acetylene flow

The relation between G and β is as follows:

G, kg/hr	10	15	25	40	65	100	150	200	400
β	3.05	2.88	2.67	2.49	2.31	2.18	2.04	1.89	1.77

4. Hoses

The inside diameter of a hose is chosen to cater for a given volume rate of gas flow. For greater strength a hose may have several plies of cotton fabric. Hoses for fuel gases are designed

for a working pressure of up to 3 atm (gauge), while oxygen hoses are made to withstand a pressure of up to 10 atm (gauge). The innermost ply should be not less than 2 mm thick, while the outer ply should be at least 1 mm thick.

A relevant Soviet standard provides for the following hose sizes:

I. D., mm	5.5	9.5	13.0
Tolerance on I. D., mm:			
Grade 1	±0.5	±0.5	±1.0
Grade 2	±0.75	±0.75	—
Number of fabric plies:			
oxygen hoses	2	3	3
acetylene hoses . . .	1	2	2

The most commonly used sizes are with an inside diameter of 9.5 mm and an outside diameter of 17.5 to 22 mm. Torches with a high gas consumption (such as used for gas-pressure welding, flame hardening, etc.) use rubberised-fabric hoses with an inside diameter of 16 and 19 mm, usually employed for pneumatic tools.

Hoses for petrol and kerosene are made from petrol-resistant rubber.

5. Manifolds

Manifolds are used for the centralised supply of oxygen and acetylene to welding stations. They are set up in a separate room, and the gas from the manifolded cylinders in conveyed to the torches by a supply piping system.

An oxygen manifold (Fig. 35) consists of two manifold headers *1* fitted with shut-off valves to which oxygen cylinders *2* are attached by means of coiled copper tubes. Each manifold has a master valve *3*. This valve is followed by a pressure gauge *4* and a manifold pressure regulator *5* from which the piping system *6* is run to the torches. The cylinders are made fast to a wall or the manifold frame to prevent them from falling. If the pressure in the shop mains is in excess of what is safe for a given job, the welding stations have separate pressure regulators in addition to the usual shut-off valves.

An acetylene manifold is similar in design to an oxygen manifold except that its header is made from a seamless steel pipe, and the manifolded cylinders are connected by flexible armoured rubberised-fabric hoses rather than by coiled copper tubes. The pressure of the acetylene discharged into the shop mains should not

exceed 1.5 atm (gauge). A medium-pressure flashback arrester must be installed where the gas is drawn off to supply a torch.

Manifolds for a discharge rate of up to 15 m³/hr may be set up in noncombustible or semicombustible locations within the
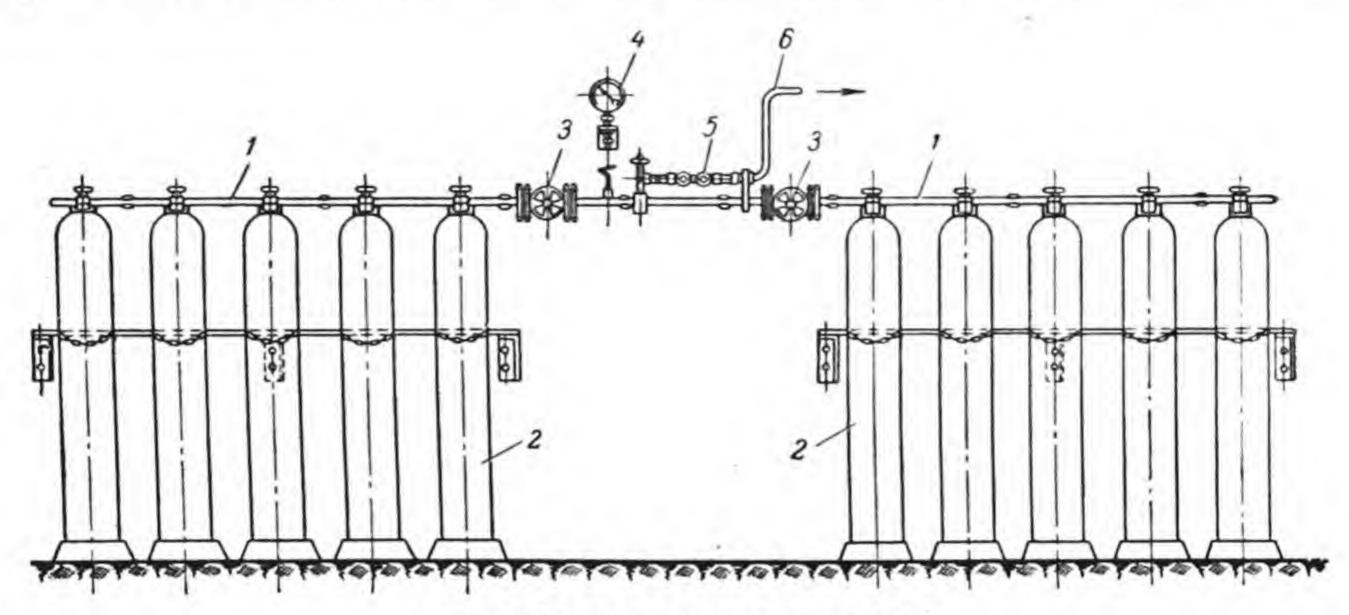
Fig. 35. An oxygen manifold

main building or in a lean-to attachment separated from the shop by a continuous masonry or ferroconcrete wall.

6. Valves

Valves for oxygen lines must be made from nonferrous metals such as brass or bronze. Before an oxygen valve may be installed on a line, it must be thoroughly degreased. Acetylene lines may use steam valves made from steel or cast iron.

Oxygen manifolds use master (manifold) shut-off valves. One design of a master valve is shown in Fig. 36a. This valve is designed to withstand a working pressure of 150 to 165 atm (gauge) and should be hydrostatically tested at 225 atm (gauge). The body *1* and pin carriage *6* of the valve are made from brass ЛС-59-1, and the tapened pin *8* is from stainless steel ЭЖ-3. Alternatively, the pin carriage may be made from aluminium-manganese bronze Бр.АМц9-2. The seal around the spindle *4* made from nitrided free-cutting steel is provided with a fibre gasket *5* clamped between the body and the gland nut *3*. Rotation of the spindle is transmitted to the pin carriage by a flat cotter *2* made from steel ЭЖ-3. The valve flanges *7* are made from steel Ст. 3 and are jointed to the valve body by a thread and subsequent brass welding or brazing with ПМЦ-51 spelter.

Fig. 36. Oxygen manifold shut-off valves:
a — flange-mounted type with a tapered valve pin; *b* — type with a babbitt disk and a female-threaded disk carriage

Fig. 37. Twin gas valve

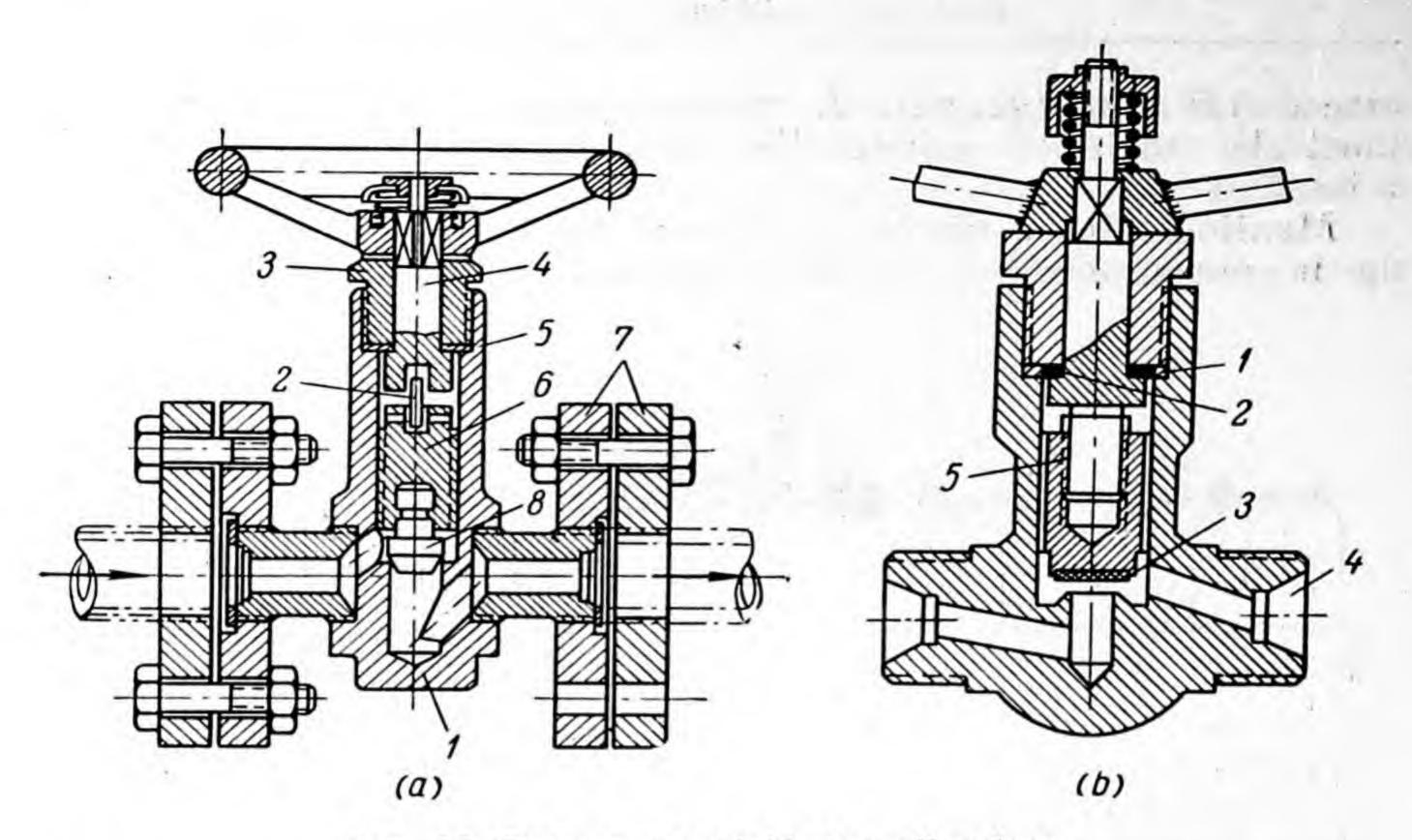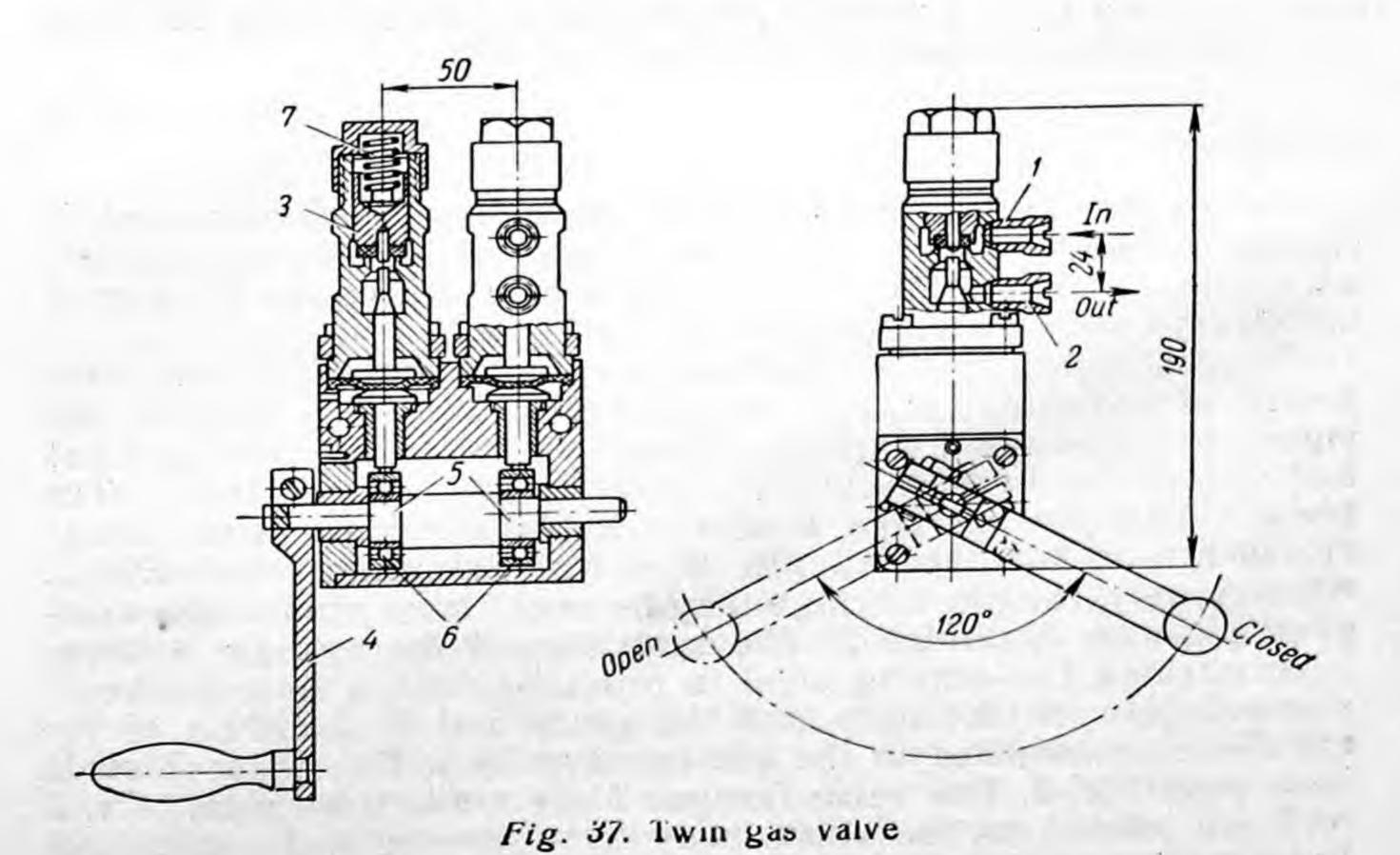Fig. 36*b* shows another design of master (manifold) valve. In this valve the gland nut has a copper sealing ring *1*. The shoulder of the spindle bears upon a flexible gasket *2* made from a fluorinated plastic material. The plug has a valve disk *3* from babbitt БН. Instead of flanges, the valve is attached to a pipeline by means of coupling nuts and is sealed by the cone *4*. The
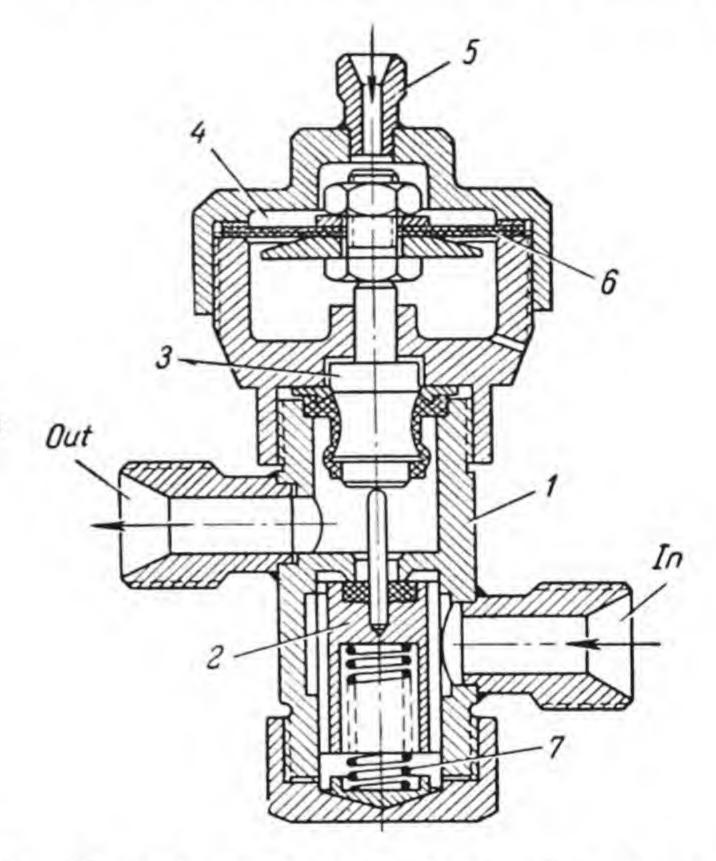
Fig. 38. Air-operated gas valve

spindle end *5* has a L. H. thread and fits into the female thread of the plug. This type of valve has proved its worth in service.

To reduce pressure losses, it is customary to install on oxygen lines station regulators with nozzles of increased flow passage. This is of primary importance in, say, cutting steel of heavy gauges, where much cutting oxygen is withdrawn at a relatively low pressure (of the order of 5 or 6 atm, gauge).

The pressure regulators used on oxygen and acetylene manifolds are designed for considerable volume rates of gas flow and relatively low pressure differentials.

In order that gas supply can be turned on or off quickly, use is made of twin gas valves. Fig. 37 shows a manually-operated twin gas valve. The gases enter the valve through the nipples *1* and issue from it through the nipples *2*. The crank *4* lifts the valve plugs *3* via the shaft *5* and ball bearings *6* eccentrically seated on the shaft. The valve plugs are closed by springs *7*. The bearings are offset relative to each other so that when the switch is moved

into the open position, the oxygen valve plug opens first, while on a closing stroke the acetylene valve plug closes first.

Remote control of gas supply is provided with air-operated or electromagnetic valves. Fig. 38 shows diagrammatically an air-operated valve. It consists of a body *1* and a valve plug *2* which is opened by a pusher *3* when the gas is to be admitted through a nipple *5* into the chamber *4* closed by a rubber diaphragm *6*. The plug *2* is closed by a spring *7* when the gas from the chamber
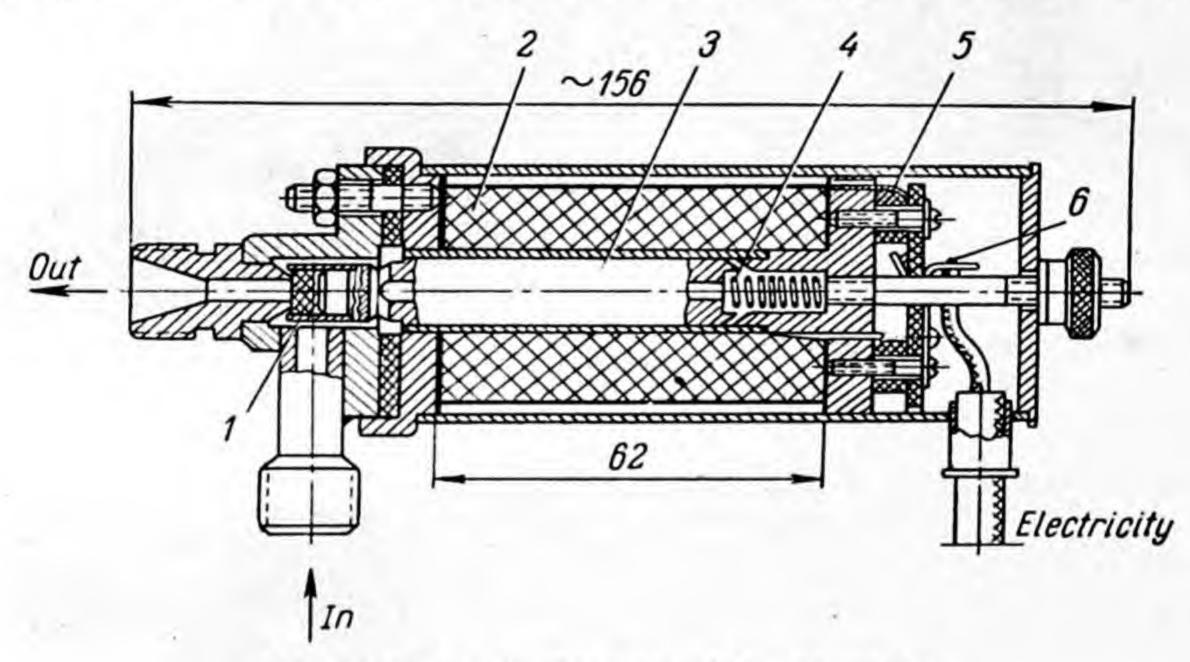
Fig. 39. An electromagnetic gas valve

4 is discharged into the atmosphere. The pressure of the gas which opens the valve must be 0.3 to 0.5 atm (gauge) higher than that of the gas passed through the valve.

An electromagnetic valve designed for a pressure of up to 12 atm (gauge) (Fig. 39) consists of a coil *2* inside which is located a steel plunger *3* terminating in a valve disk and a disk carriage *1*. When the coil is energised, the plunger is pulled into the coil, and the carriage moves the valve disk away to open the nozzle. When the coil is de-energised, the opposing spring *4* forces the disk carriage to close the nozzle. The body of the valve has a block *5* with terminals *6* with which the coil is connected to a switch on the control panel.

The pressure loss in valves may be determined by the following equation:

$$\Delta p = K_\Delta \frac{V^2}{p_{av}}, \qquad\qquad\text{(VI.6)}$$

where Δp = pressure loss, atm;
V = gas flow, m³/hr;

p_{av} = average gas pressure across the valve, atm (abs); for $p_{av} < 15$ atm (abs), it may be assumed that $p_{av} = p_g + 1$;

p_{th} = pressure before the throat, determining the flow through the valve, kg/cm²;

K_Δ = coefficient of friction.

According to the experimental data obtained by S. G. Guzov, the values of K_Δ for oxygen at a flow of 30 m³/hr are as follows:

	Values of K_Δ
Type PK-47 cylinder regulators	11×10^{-3}
Type PK-50 cylinder regulators	2.6×10^{-3}
Type PKП-52 station regulators	0.9×10^{-3}
Hoses 9.5 mm I. D., 10 m long	3×10^{-3}
Type УР-48 and УРЗ-49 manual cutting torches	12×10^{-3}
Type РМ-3Г cutting machines	8×10^{-3}
Type РМР-Ш and СГ-2 cutting machines	14×10^{-3}
Type РВП-47 flame-machining torches	13×10^{-3}
Type К-48 kerosene cutters	28×10^{-3}
Type К-51 kerosene cutters	9×10^{-3}

7. Flow Indicators

Both field operators and researchers of flame welding and cutting equipment mostly use float-type flow indicators, known as rotameters.

A rotameter (Fig. 40) consists of a graduated glass tube *1* inside which is positioned a float *2* made from aluminium, hard rubber or stainless steel. The flow is restricted by the annular area between the float and the tube which tapers out towards its top. With the fluid flowing up, the float rises until the annular area between the float and the tube becomes sufficiently large to pass a given volume of fluid at a pressure balancing the weight of the float at a given float level.

Thus, the rotameter operates at a fixed pressure differential, with the fluid flowing through a variable-area orifice. Each division of the scale etched on the surface of the glass tube corresponds to a difinite flow rate at a given moment of time. For the same volume of gas to pass through the orifice at a given pressure differential, the annular area will have to be greater (the float will be positioned higher in the tube) for a gas with a lower density.

8*
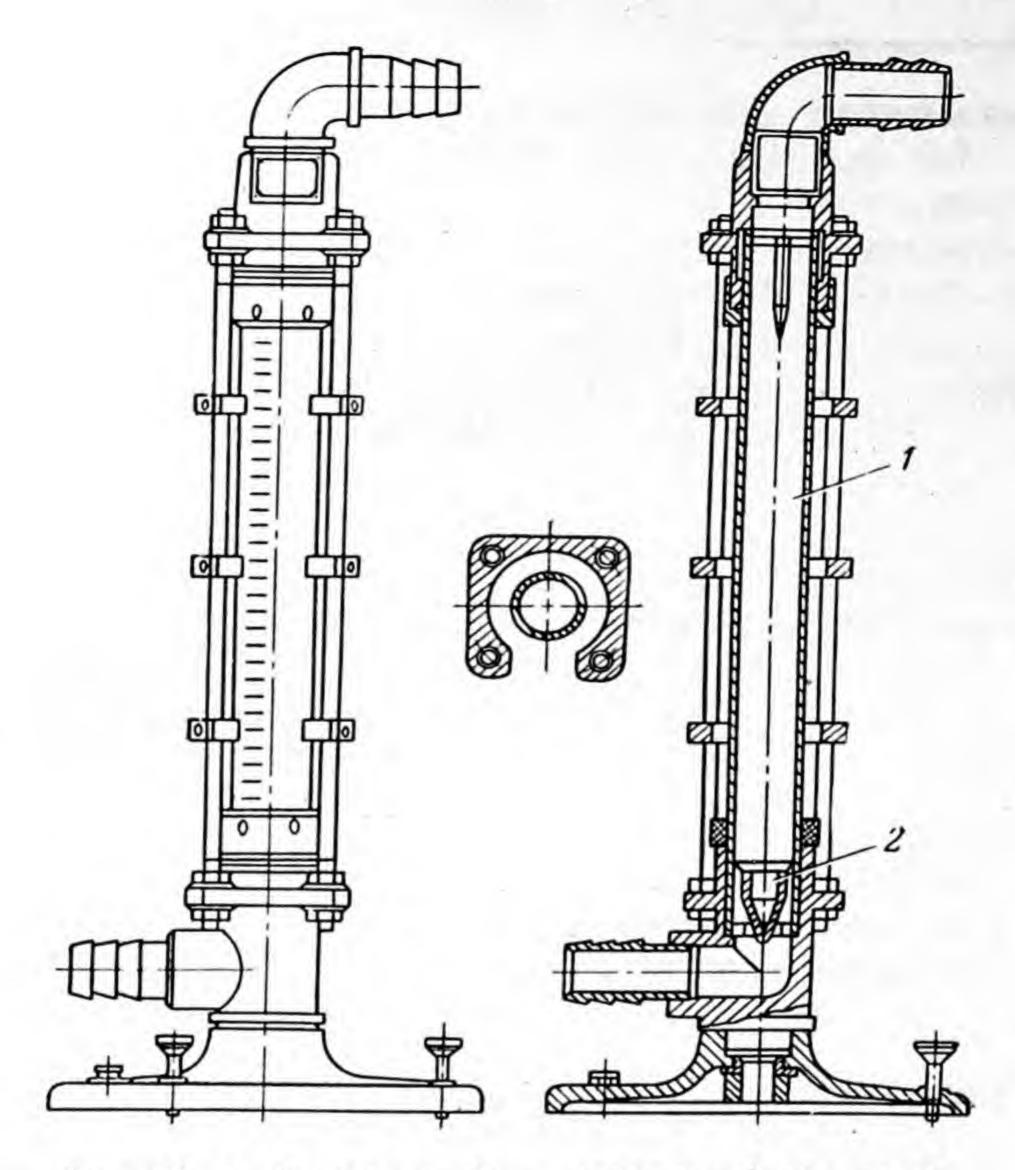
Fig. 40. A float-type flow indicator (rotameter)
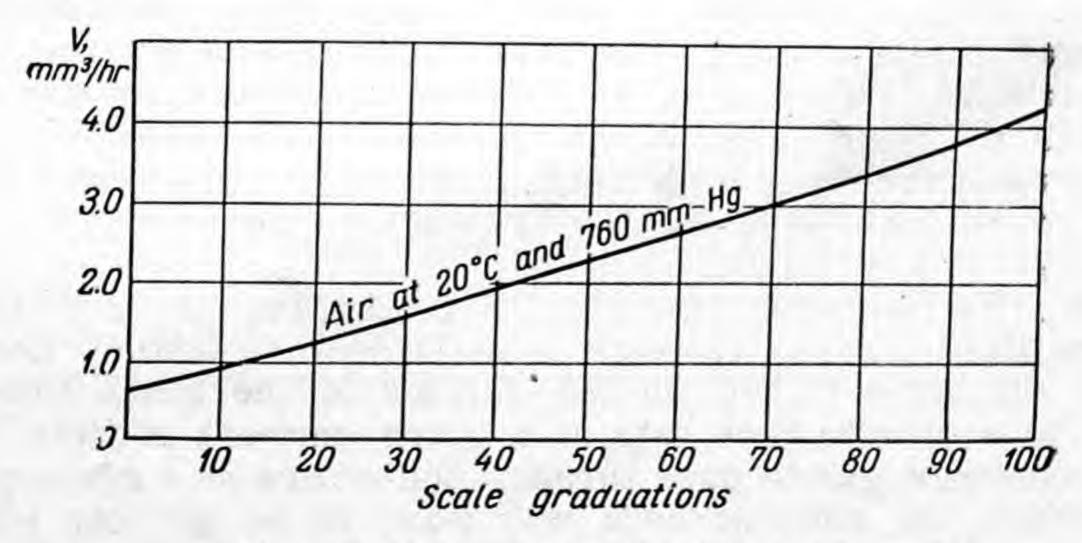
Fig. 41. Rotameter calibration chart

The equation for volumetric flow through a rotameter is:

$$V_g = V_r \sqrt{\frac{\gamma_r}{\gamma_g} \frac{T_r}{T_g} \frac{p_g}{p_r}}, \qquad (VI.7)$$

where V_g = volumetric flow of the gas under measurement (at 20° C and 760 mm Hg), m³/hr;

V_r = volumetric flow from the calibration chart of the rotameter for a given actual reading and under the conditions of rotameter calibration (γ_r, T_r, and p_r), m³/hr;

γ_g and γ_r = specific weight of the gas under measurement and of the gas used for rotameter calibration, respectively, kg/m³;

p_g and p_r = absolute pressure of the gas under measurement and of the gas used for rotameter calibration, respectively, before the rotameter, atm (abs);

T_g and T_r = absolute temperature of the gas under measurement and of the gas used for rotameter calibration, respectively, deg. K.

Fig. 41 shows the calibration chart of a rotameter. Manufacturers usually calibrate rotameters with air. Therefore, when they are used on other gases, their readings have to be adjusted by means of Eq. (VI. 7) and the calibration chart attached to the certificate of each instrument.

The oxy-gas flame

COMBUSTION AND THE CHEMICAL PROPERTIES OF THE FLAME

1. Combustion and the Structure of the Flame

Combustion is a rapid reaction accompanied by the evolution of so much heat that the gaseous products of combustion are heated to incandescence.

In most cases combustion is the process of chemical combination of solid, liquid and gaseous substances with oxygen. As applied to metals, a more appropriate name would be intensive oxidation.

Many metals can also burn with chlorine and fluorine. Iron and copper can burn with sulphur vapours, while sodium or barium oxides with carbon dioxide.

To a flame-process engineer the principal interest in connection with combustion lies in various mixtures of gases or vaporised liquids and oxygen or air. A gas mixture ignites at a temperature which varies with the conditions obtaining at the moment. * As soon as the mixture has ignited, any heating from the outside becomes unnecessary, as enough heat is now being evolved to sustain the burning of the remaining mixture. For combustion to be truly steady, enough heat must be given up both to raise the temperature of the remaining gas mixture and to make up for the losses of heat to the surroundings. In tubes of small diameter, especially in capillary tubes, the burning of a gas is impossible to sustain owing to the great heat abstraction through the walls and the high ratio of the surface area to the volume of the orifice.

The burning of a fuel gas in oxygen or air is determined by what is known as the limits of flammability (see Table 4).

On the basis of flame velocity, combustion may be classed into three types:

(1) quiet burning, with a flame velocity of not over 10 or 15 m/sec;

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* According to a current theory, the point at which a combustible gaseous mixture ignites is not a physical constant, but varies with the conditions of combustion.

(2) deflagration, with a flame velocity of several hundred metres per second;

(3) detonation, with a flame velocity of over 1,000 m/sec.

Flame velocity is mainly dependent on the composition and pressure of the gas mixture, the space where the mixture is contained and the thermomechanical conditions on its boundary (for a mixture burning in a tube the main variable is the diameter of the tube). * For each gas-oxygen mixture, the maximum flame velocity is attained at a definite gas-to-oxygen ratio. An excess of oxygen *above this ratio* reduces the flame velocity and has a retarding effect on combustion, as it does not take part in burning.

Flame velocity is also dependent on the purity of a combustible gas, and is reduced as the content of noncombustible gases in it increases.

An increase in the pressure of a gas mixture increases its flame velocity, as does the increased diameter of the tube in which the gas mixture is burned.

The combustible gases or vaporised liquids employed for the welding or heating of metals are mainly hydrocarbons or mixtures of hydrocarbons and other gases, such as CO or CO_2. Of all fuel gases, only hydrogen is used in its pure form.

For a composite fuel gas the maximum flame velocity can be determined by the equation

$$u_m = \frac{\dfrac{r_1}{R_1} u_1 + \dfrac{r_2}{R_2} u_2 + \cdots + \dfrac{r_n}{R_n} u_n}{\dfrac{r_1}{R_1} + \dfrac{r_2}{R_2} + \cdots + \dfrac{r_n}{R_n}}, \qquad (VII.1)$$

where u_m = maximum flame velocity for a given gas-oxygen mixture, cm/sec;

$r_1, r_2, \ldots, r_n$ = proportion of constituent gases in gas-oxygen mixture (per cent by volume);

$R_1, R_2, \ldots, R_n$ = proportion of constituent gases in gas-oxygen mixture (per cent by volume) at maximum flame velocity;

$u_1, u_2, \ldots, u_n$ = respective maximum flame velocities for the constituent gases when burned with oxygen, cm/sec.

On the strength of Fig. 69 and taking into account the effect of the inert part B (ballast) of the combustible gases, Eq. (VII. 1)

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* Most often the flame velocity of inflammable mixtures is determined by measuring the velocity of the flame front in a tube filled with the mixture under test.
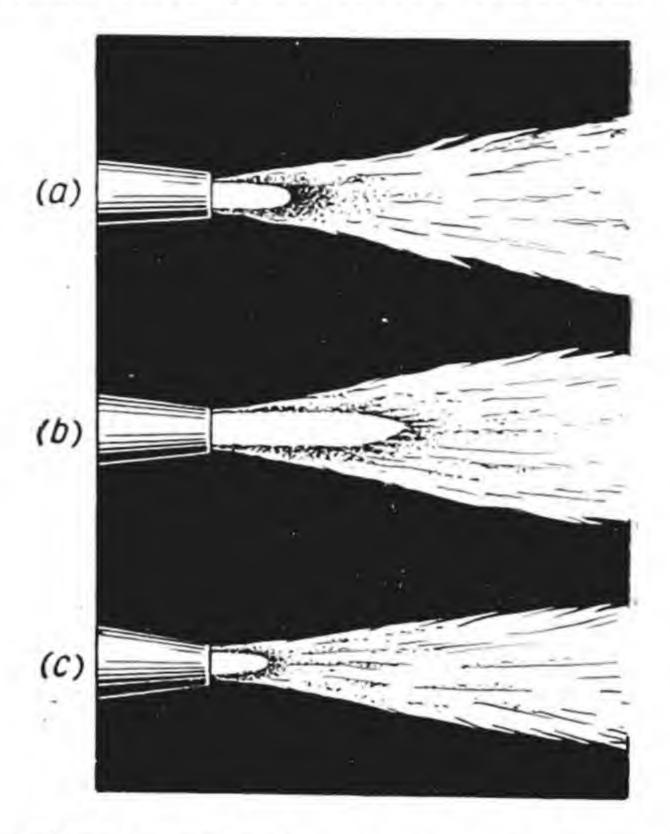
may be rewritten as follows:

$$u_m = \frac{131.7 r'_{H_2} + 110 r'_{CH_4} + 329 r'_{C_3H_8} + 15.8 r'_{CO}}{14.8 r'_{H_2} + 33.3 r'_{CH_4} + 89 r'_{C_3H_8} + 14.4 r'_{CO}} - 0.05 B, \quad (VII.2)$$

where r' is the proportion of the individual gases in the mixture (in per cent by volume) of the combustible part of the mixture.

Fig. 42. Chart showing three types of oxy-gas flame:

a — neutral or balanced flame; b — carburising flame; c — oxidising flame

Eq. (VII. 2) applies where the ballast content in a gas is less than 35 per cent.

The oxy-gas welding flame consists of three regions or zones: (1) the inner luminous cone; (2) the intermediate (reducing) zone; (3) the flame envelope or streamer (the oxidising zone).

All combustible gases containing hydrocarbons produce a flame with the luminous inner cone, similar in structure to the oxy-acetylene flame (Fig. 42). The cone grows better defined as the content of hydrocarbons in the gas increases. Apart from acetylene, the other combustible gases producing a flame with a luminous inner cone are methane, natural gas, retort gas and refinery oil gas.

As distinct from hydrocarbon gases, the oxy-hydrogen flame has no luminous inner cone and has a light-yellow tinge. The absence of a luminous inner cone (which, in other gases, is due to their carbon) makes it difficult for the welder to adjust the flame by its appearance.

The further discussion will be concentrated on the oxy-acetylene flame, as acetylene is the most commonly used fuel gas in welding and cutting.

It is customary to class an oxy-acetylene flame as neutral or balanced, oxidising, and carburising or reducing, depending on the proportion of oxygen and acetylene in the mixture burned.

The neutral or balanced flame has a well defined luminous cone gradually rounding off towards the tip (Fig. 42a). It is produced by burning an approximately one-to-one mixture of oxygen and acetylene.

The carburising or reducing flame is obtained by burning an excess of acetylene. In this case the rate of oxidation is slowed down, more acetylene is burned with the oxygen from the surrounding air, and the flame grows in size. The inner cone is not so well defined as in the neutral flame, and there is no sharp delineation between the outer flame envelope and the intermediate or excess acetylene zone (Fig. 42b). The outer flame takes on a reddish tinge, and some soot depending on the amount of excess acetylene is produced. This flame is sometimes called excess acetylene, or reducing, as well as carburising.

The oxidising flame is produced by turning on excess oxygen. The white cone is very short, and not so well defined. The intermediate cone and the envelope are also short due to intensive oxidation (Fig. 42c). The streamer of the flame is blue. The oxidising flame produces a specific sound which grows louder as the pressure and proportion of oxygen in the mixture increase.

The complete combustion of acetylene with oxygen is described by Eq. (II.11). For the flame produced at the tip of a welding torch the reaction actually proceeds in two stages which may be presented to a first approximation as follows:

(1) combustion due to the oxygen supplied by the torch (or primary combustion):

$$C_2H_2 + O_2 = 2CO + H_2 + 107.58 \text{ kcal/g-mole;} \qquad \text{(VII.3}$$

(2) combustion due to the oxygen of the air (or secondary combustion):

$$2CO + H_2 + 1.5O_2 = 2CO_2 + H_2O + 203.57 \text{ kcal/g-mole. (VII.4)}$$

From Eq. (VII.3) it follows that for the acetylene to burn to CO and H_2 there should be one part acetylene to one part oxygen, by volume (V_a and V_{ox}, respectively). This is what is usually called the balanced flame. In practice, however, some of the hydrogen burns to water vapour due to the oxygen of the combustible mixture; nor is the oxygen too pure. Therefore, it takes

more than one part oxygen to obtain the balanced flame:

$$\beta_0 = \frac{V_{ox}}{V_a} = 1.1 \text{ to } 1.2.$$

In the case of an oxidising flame, especially when $\beta_0 = 1.5$ the primary and secondary stages of combustion proceed according to the following reactions:

$$C_2H_2 + 1.5O_2 = 2CO + H_2 + 0.5O_2 \qquad (VII.5)$$

and

$$2CO + H_2 + 0.5O_2 + O_2 = 2CO_2 + H_2O. \qquad (VII.6)$$

The intermediate zone of this flame carries some 6 or 7 per cent oxygen and about 5 per cent carbon dioxide (VII.3). This zone is no longer of reducing nature and becomes oxidising instead.

In the case of a carburising flame, especially when $\beta_0 = 0.5$, the primary and secondary reactions will be as follows:

$$C_2H_2 + 0.5O_2 = CO + C + H_2 \qquad (VII.7)$$

and

$$CO + C + H_2 + 2O_2 = 2CO_2 + H_2O. \qquad (VII.8)$$

The intermediate cone now carries a considerable amount of free carbon, and it becomes carburising.

When *too much* acetylene is present in the mixture, some free carbon will also be found in the streamer.

Reactions (VII.3) through (VII.8) do not give a complete picture of combustion, as the fuel undergoes some intermediate transformations.

The contemporary theory of combustion, as developed by N. N. Semyonov of the U.S.S.R. and his coworkers, especially as far as gaseous hydrocarbons are concerned, differentiates the following basic stages in the combustion of the oxy-acetylene mixture.

The induction stage (which takes place in the inner cone) is characterised by the thermal decomposition of the acetylene in equilibrium with the oxygen. In the absence of oxygen, at 800 to 1,250° C the fuel would break up into its components C and H_2 with the possibility of methane (CH_4) existing as an intermediate product. For hydrocarbons of the C_xH_y type this decomposition in the absence of oxygen would proceed as follows:

$$C_xH_y \rightleftarrows xC + \frac{y}{2} H_2. \qquad (VII.9)$$

For acetylene:

$$C_2H_2 = 2C + H_2. \qquad (VII.10)$$

At above 800° C, however, the decomposition may be accompanied by the formation of methane which is more stable thermally than acetylene:

$$2C_2H_2 = CH_4 + 3C. \qquad (VII.11)$$

At above 1,000° C the methane will, however, break up as follows:

$$CH_4 = C + 2H_2. \qquad (VII.12)$$

In the presence of oxygen, acetylene begins to decompose at a lower temperature; the rate of decomposition rises due to the oxidation of the products of decomposition, which, when oxidised, are less thermally stable.

When acetylene is decomposed in the presence of oxygen, the final products will be carbon monoxide and hydrogen.

Being an unsaturated hydrocarbon, acetylene readily decomposes and combines with oxygen. The products of the reaction, however small their quantity may be, speed up the rate of decomposition. In the presence of oxygen the decomposition of acetylene is explained by a current hypothesis as follows:

$$
\left.
\begin{aligned}
&C_2H_2 + O = C_2H_2O; \\
&C_2H_2O + O_2 = C_2H_2O_2 + O; \\
&C_2H_2O_2 \rightarrow HCHO + CO \\
& \downarrow \\
& H_2 + CO
\end{aligned}
\right\} \qquad (VII.13)
$$

This chain of reactions results in CO and H_2 — much more stable products. In the actual decomposition, however, some of the acetylene is completely broken up into its components to produce free carbon in the form of fine specks of soot, making up a thin white-hot envelope around the inner cone. With an excess of acetylene, the amount of free carbon sharply increases; the inner cone grows in size, and the white-hot envelope of soot particles expands. As more acetylene becomes available, the intermediate zone of the flame, containing the products of the thermal decomposition (CO and H_2), disappears completely, and the flame produces much carbon black.

It should be noted, however, that we still lack an exact knowledge of the reactions that take place during the thermal decomposition of acetylene. The spectrograms of the inner cone show the presence of such radicals as CH, OH and HCO, as well as of H, C_2 and O_2.

The rate and nature of the thermal decomposition of acetylene during the induction stage depend on the temperature, pressure

and composition of the mixture as well as on the ratio of the surface area to the volume of the gas.
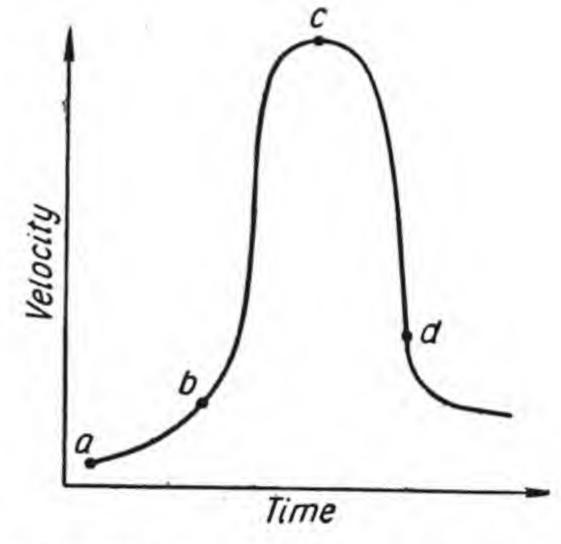
The duration of the induction stage can be shortened by raising the temperature of the mixture and reducing its pressure. The decomposition can also be speeded up by making more oxygen available in the mixture.

During the induction stage the rate of decomposition increases slowly, as can be seen from reference to the leg *"a-b"* of the curve in Fig. 43.

Fig. 43. Flame velocity vs. time

The oxidation reactions during the induction stage are low in intensity, and the action of oxygen is not enough to speed up the process.

The ignition stage (which takes place in the middle zone) is shown by the leg *"b-c"* in Fig. 43. In this stage the oxidation reactions go on at a higher rate, and the carbon monoxide and hydrogen are burned to carbon dioxide and water vapour. The ignition point is at *"c"*.

The final stage (which takes place in the streamer) is shown by the leg *"c-d"* in Fig. 43. During this stage, the products of acetylene dissociation are oxidised to CO_2 and H_2O:

$$CO + O \rightleftarrows CO_2; \qquad (VII.14)$$

$$H_2 + O \rightleftarrows H_2O. \qquad (VII.15)$$

During this stage and past the point *"c"* in Fig. 43 the rate of combustion is sharply reduced. The position of the point *"b"* in Fig. 43, i. e., the completion of the induction stage, depends on the temperature of the mixture. Fig. 44 shows how its temperature varies during the combustion process. During the induction stage (in the inner cone) the temperature of the mixture rises from T_0 to the ignition point T_{ign}. According to a contemporary hypothesis of flame propagation in a combustible mixture

freely issuing from an orifice, there is, in front of the flame envelope, a very thin layer (usually under 1 mm) of mixture, on the boundary of which the temperature is already increased due to heat conduction from the flame envelope. In fact, it is in this layer that the temperature of the combustible mixture rises from T_0 to T_{ign}. During the ignition stage (in the intermediate zone),
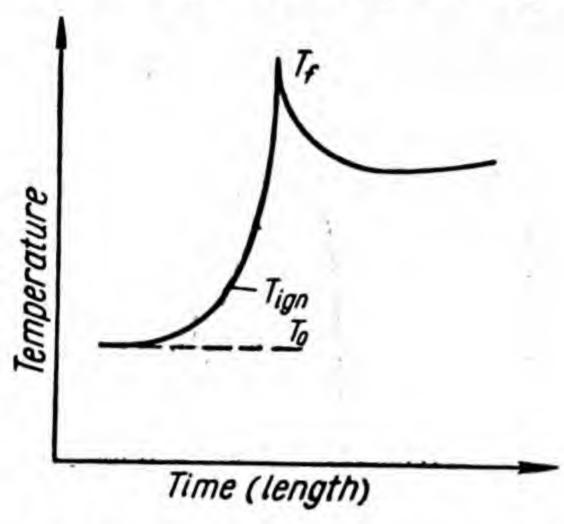
Fig. 44. Changes in the temperature of the air-fuel mixture during combustion

the temperature sharply rises from the ignition point T_{ign} to the maximum flame temperature T_f. During the final stage (in the streamer) the temperature falls.

2. Chemical Properties of the Oxy-acetylene Flame

The chemical composition of the flame varies across its volume. Above all, it depends on the proportion of the gas and oxygen in the mixture and on how much clearance air is present in the flame. From the viewpoint of welding, the intermediate zone, which is next to the inner cone, is of the greatest interest.

The chemical composition of the flame can be found by computation, by chemical analyses and spectroscopy.

The chemical analyses of the flame obtained by a number of investigators are not too accurate, as the samples drawn from the various parts of the flame may change their gas composition during the subsequent cooling.

Chemical analysis is best applicable to the zones of the oxy-acetylene flame which are within some distance of the inner cone. As to the inner cone which carries the unstable intermediate products of acetylene dissociation, the most accurate results are obtained with spectroscopy.

Both computational and analytical determinations make it possible to draw the following conclusions with respect to the oxy-acetylene flame:

1. When the ratio of oxygen to acetylene in the mixture is anywhere between 1.1 : 1 and 1.2 : 1, the intermediate zone carries the maximum amount of CO (60 to 66 per cent) and H_2 (34 to 40 per cent).

2. As the oxygen-to-acetylene ratio increases, the amount of CO and H_2 in the intermediate zone is reduced, and that of CO_2 and H_2O is increased.

3. The flame contains more atomic than molecular hydrogen.

4. As we move away from the inner cone both along and across the flame, the amount of CO and H_2 desreases, and that of CO_2, H_2O and N_2 increases.

5. The intermediate (reducing) zone of the flame is but a few millimetres long in the neutral flame ($\beta_0 = 1.1$-1.2), and becomes shorter with an excess of oxygen.

The theoretical basis of flame adjustment has been worked out by A. N. Shashkov who has established general principles for the determination of optimum gas-oxygen mixtures for a variety of metals and alloys. Prior to his findings, some investigators had classed the oxy-acetylene (welding) flame incorrectly into oxidising, neutral and carburising. The term "neutral" is irrelevant, as a truly neutral flame would neither oxidise nor reduce the metal, as it would be in equilibrium with both the metal and its lowest oxide. On the other hand, though the "neutral" flame employed in welding (with an oxygen-to-acetylene ratio of 1.1-1.2) strongly resists oxidation, in some cases (when welding iron, copper and nickel) it may reduce the metal of the molten pool owing to the presence of an appreciable amount of CO and H_2 in the intermediate zone.

Referring to the equilibrium diagrams of H_2 and CO with iron oxide and iron (Figs 45 and 46), only those mixtures will produce a neutral flame which are on the lines of equilibrium. Obviously, the chance for a welding operator to obtain a truly neutral flame is very remote, indeed. Therefore, the term "normal" seems more appropriate.

The lower limit of β_0 is determined by the oxygen required to oxidise all of the carbon to CO. The combining proportion (by volume) can be found from the reaction (VII. 3) and is unity. With the lower ratio, the mixture will have an excess of free carbon which will burn with the oxygen of the air and form another cone capable of reducing the metal in welding.

The upper limit of β_0 for the normal flame must be such as not to upset the equilibrium between CO and H_2, on the one hand, and FeO, on the other. On the basis of the respective equilibrium diagrams (Figs 45 and 46) it has been established analytically that this ratio must be 1.3 : 1. With more oxygen in the flame,

an oxide film appears on the surface of the molten pool, thereby indicating that the flame has changed from normal to oxidising.
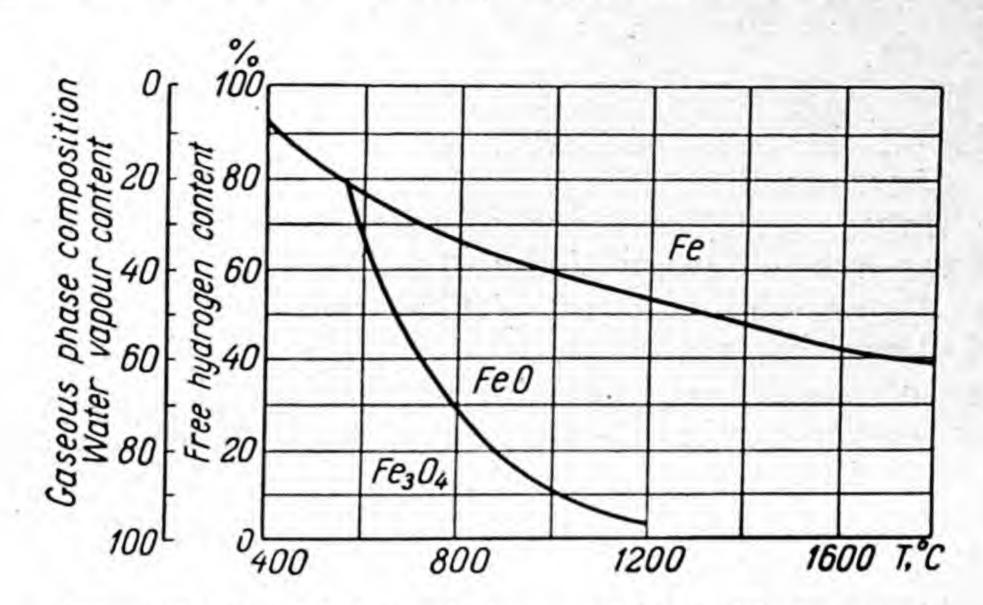
Fig. 45. Equilibrium diagram of H_2 with iron and iron oxides

The actual amount of oxygen in the normal flame is a function of many variables, the principal ones being the metal to be
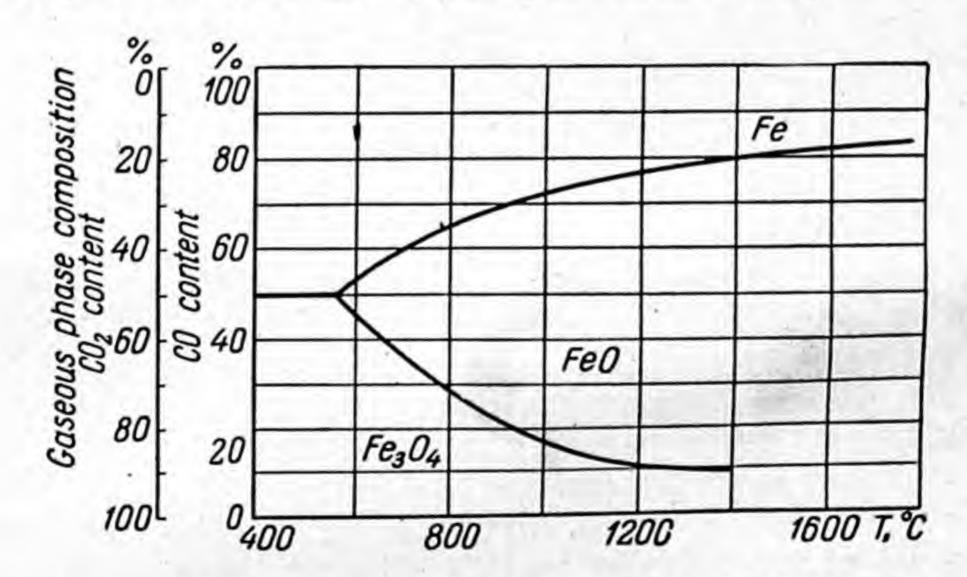
Fig. 46. Equilibrium diagram of CO with iron and iron oxides

welded, the temperature of the molten pool, and the composition of the fuel gas. Before this problem can be solved for low-carbon steel, the conditions should be examined under which iron oxide is formed and reduced in welding according to the reaction:

$$2Fe + O_2 = 2FeO + 128.6 \text{ kcal/g-mole.} \qquad (VII.16)$$
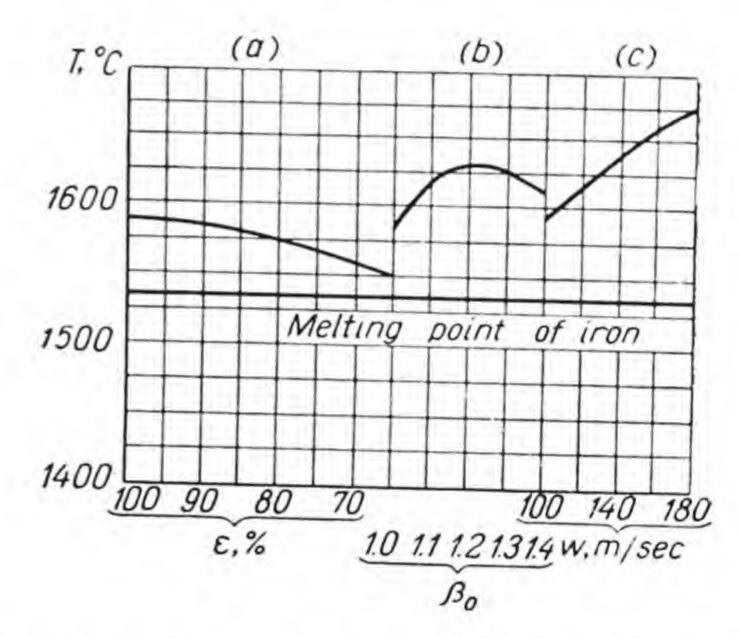
Although in the case of solutions, only two phases (the liquid or solid solution of FeO in Fe, and one gaseous phase) can take part in this reaction, it will not be inappropriate to assume, for simplicity, that three phases (two liquid or solid phases and one gaseous phase) are involved in the reaction. Then this reversible reaction will only have one degree of freedom, and for its equilibrium it is necessary that

$$K_{eqFeO} = p_{O_2}, \qquad\qquad (VII.17)$$

from which it follows that at a given temperature ferrous oxide can coexist with pure iron only at a definite partial pressure of oxygen above them, or p_{O_2}.

Fig. 47. Temperature of the welding pool as a function of (*a*) oxygen purity ε; (*b*) oxygen-to-acetylene ratio β_0; and (*c*) average velocity of the mixture, w

From Fig. 47 showing the temperature of the molten pool, as measured pyrometrically, it can be seen that depending on the purity of oxygen, the oxygen-gas ratio and the average velocity of the mixture it may vary between 1550° C and 1675° C.

The average temperature, corresponding to $\beta_0 = 1\text{-}1.1$ under normal conditions of welding, may be taken equal to 1600° C.

Fig. 48 presents the dissociation pressures of ferrous oxide at the above temperatures. On the basis of these pressures it is possible to class the gaseous phase by its action into oxidising and reducing. In the oxidising gaseous phase, the pressure of the oxygen is higher than that of the ferrous oxide, and the oxygen is continually transferred from the gaseous phase into the iron until the oxygen is depleted or all of the iron is oxidised to ferrous oxide. In the reducing phase, the pressure of oxygen is lower than that of the ferrous oxide, and the reducing gaseous phase will absorb it until its excess in the iron is depleted or all of the ferrous oxide is reduced (for a three-phase system).

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For the welding flame where the reaction proceeds in a mixture of oxygen and other gases not taking part in it, the direction of the reaction will be governed by the partial pressure of oxygen.
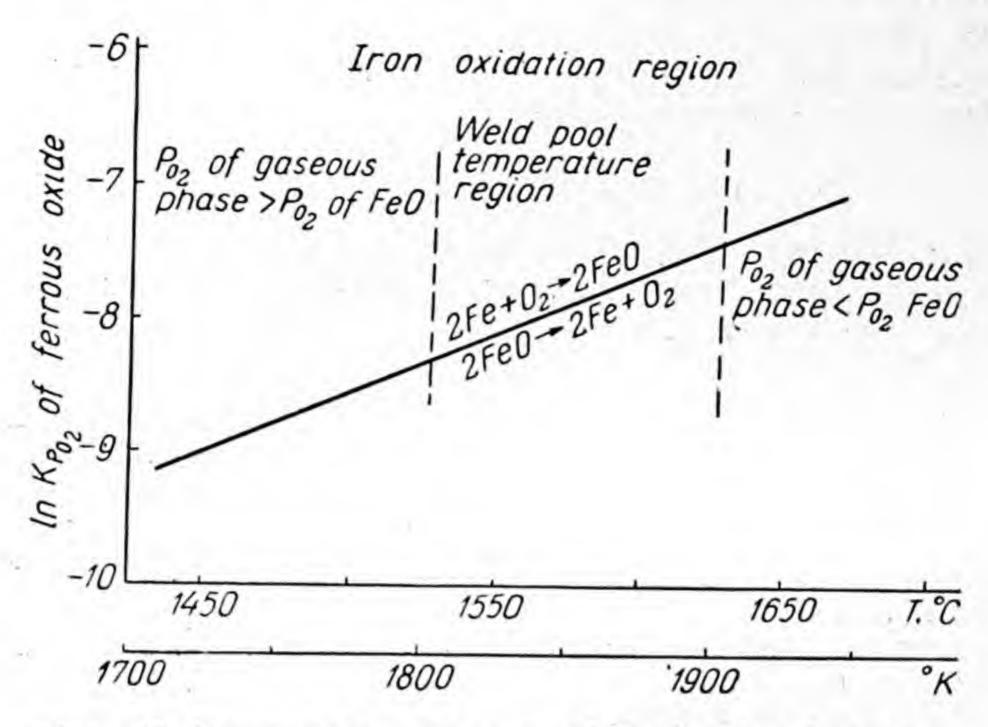
Fig. 48. Dissociation pressure of FeO related to temperature

Hydrocarbons burn in the welding flame according to reactions (II. 13). These are reversible reactions which, by the phase rule, have three degrees of freedom or variance (concentration, temperature and pressure). The concentration ratios of the various components of the system can be determined by equilibrium equations according to the Law of Mass Action:

$$K_{eq H_2O} = \frac{p_{H_2}^2 p_{O_2}}{p_{H_2O}^2} \; ; \quad K_{eq\, CO} = \frac{p_{O_2}}{p_{CO_2}^2} \; ; \quad K_{eq\, CO_2} = \frac{p_{CO_2}^2 p_{O_2}}{p_{CO_2}} . \quad (VII.18)$$

THE THERMAL PROPERTIES OF THE OXY-ACETYLENE FLAME

1. Flame Temperature

Flame temperature has a decisive effect on all of its other characteristics. As flame temperature rises, the metal is heated and melted more efficiently.

Flame temperature is not constant throughout the flame — it varies with the flame composition both axially and radially. For the majority of gaseous hydrocarbons which produce an inner cone when burned with oxygen, the highest temperature is in the intermediate zone, next to the inner cone.
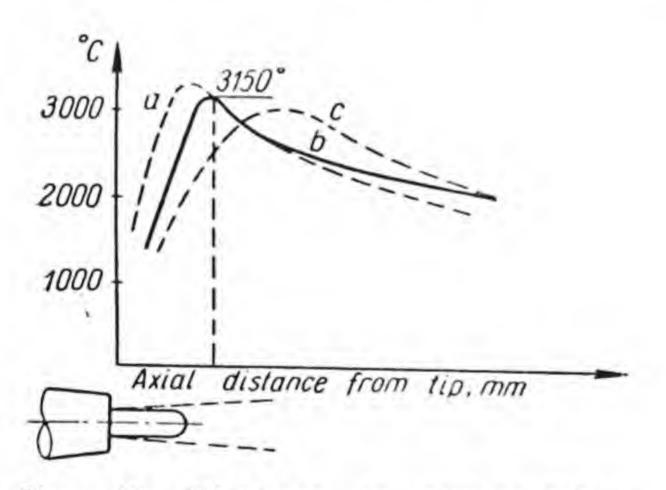
Fig. 49. Changes in oxy-acetylene flame temperature along the flame axis in (a) an oxidising flame; (b) a normal flame and (c) a carburising flame
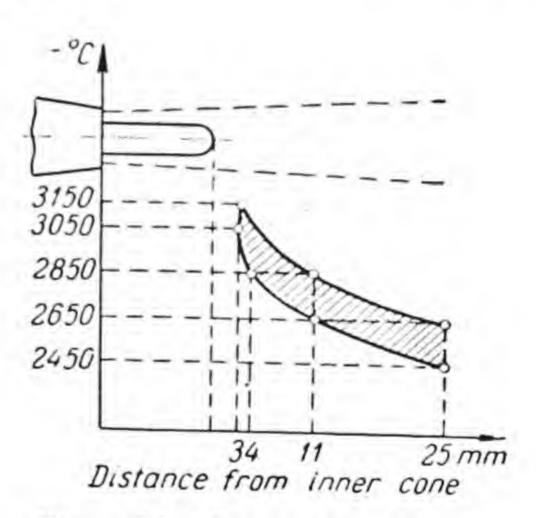
Fig. 50. Temperature gradient along the oxy-acetylene flame

Naturally, welding must be done in this zone, the more so that the intermediate zone has a reducing effect owing to the presence of carbon monoxide and hydrogen. Accordingly, the torch should be positioned so that the inner cone is within 2 or 3 mm of the metal surface.

Flame temperature is also dependent on the oxygen-to-acetylene ratio, or $\beta_0 = O_2/C_2H_2$. Increasing the ratio to a certain point

raises the temperature of the flame. Investigators are far from being unanimous as to how great this ratio should be in order to obtain the highest flame temperature. The limits, however, are from 1.2 to 1.9 for the oxygen-to-acetylene ratio and from 3100 to 3160° C for flame temperature.
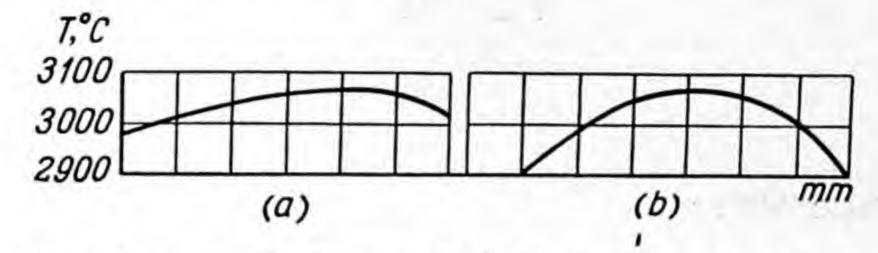
Fig. 51. Temperature gradient across the flame:
a — within 15 mm of the tip; b — within 20 mm of the tip

Fig. 49 shows how the temperature varies along a normal, an oxidising and a carburising oxy-acetylene flame. As can be seen, the oxidising flame has the highest maximum temperature, and the carburising flame, the lowest.
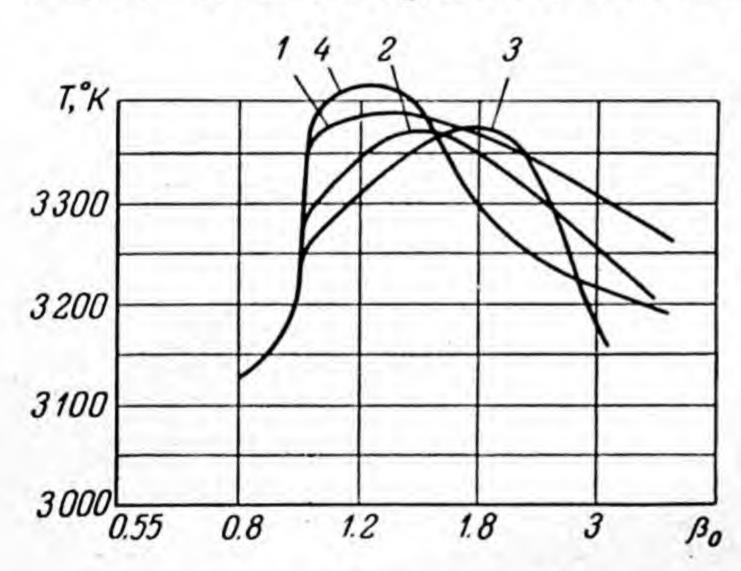
Fig. 52. The temperature of the oxy-acetylene flame as a function of oxygen-to-acetylene ratio (β_0), according to various investigators: curves 1 and 4 are experimental; curves 2 and 3 are computed

Increasing the proportion of acetylene in the mixture moves the hottest zone of the flame further away from the tip, because flame velocity decreases, the flame grows smaller in size and the heat becomes less concentrated.

Flame temperature can be computed on the basis of thermodynamics or measured directly by, say, the spectrometric method which yields the most accurate results.

Fig. 50 shows the temperature of the oxy-acetylene flame as measured by the spectrum line conversion method for an oxygen-to-acetylene ratio of 1.1-1.2 and a volume rate of acetylene flow of 500 litres per hour.

Referring to Fig. 51, it will be seen that the temperature varies both along and across the flame.

Generally, as has been noted elsewhere, flame temperature is substantially dependent on the oxygen-to-gas ratio, reaching its peak with an excess of oxygen.

Fig. 52 shows the measured and computed maximum temperature of the oxy-acetylene flame for various oxy-gas mixtures.

2. Heat Transfer from the Flame to the Metal

The flame is a local source of heat. It raises the temperature of the metal through heat transfer due to forced convection and radiation. Radiation accounts for a negligible proportion of the overall heat transfer (some 5 to 10 per cent). Therefore, the oxy-acetylene flame may, to a first approximation, be regarded as a convective source of heat. The rate of forced convective heat transfer is mainly dependent on the difference in temperature between the flame and metal and the velocity at which the gases travel relative to the metal surface, increasing with both factors. For the oxy-acetylene flame the rate is greater than for any other fuel gas owing to a much greater flame temperature (3100 to 3150° C in the intermediate zone of a normal flame) and the higher velocity of the hot gases (several ten metres per second).

In general, the quantity of heat imparted to unit area per unit time by a flame, or its specific heat flux, q_2, is in accordance with Newton's law of cooling:

$$q_2 = \alpha (T_g - T), \qquad (VIII.1)$$

where q_2 is the specific heat flux in cal/cm²-sec;
α is the coefficient of heat transfer from flame to metal in cal/cm² per sec per deg. C equal to the sum of the coefficients of convective heat transfer, α_c, and of radiant heat transfer, α_r;
T_g is the temperature, deg. C, of the hot gases;
T is the temperature, deg. C, of the metal surface.

The specific heat flux, q_2, increases with the coefficient of heat transfer, α, and the difference in temperature between flame and metal.

That Eq. (VIII.1) is a formal expression can be seen from the fact that the coefficient of heat transfer, α, is in turn dependent on the temperature of the flame and metal surface, decreasing as the temperature of the metal surface is raised.

There exists, at present, no method for determining the coefficient of heat transfer from flame to metal analytically, as we lack experimental data which would relate it to the variables of

the process. Reliance must therefore be put on the values of α obtained by correlation of the temperatures measured for the metal heated by the oxy-acetylene flame with the computed data on the heat transfer for distributed sources. Credit for this is due to N. N. Rykalin, Corresponding Member of the Soviet Academy of Sciences.

3. Distribution of the Specific Heat Flux

As it comes in contact with the metal surface, the flame is deflected so that it covers a considerable area. This area is usually called a *hot spot*. The shape and size of the hot spot vary from work to work, and depend on the shape and size of the flame and the angle it makes with the metal surface.

(1) The Flame of Simple Torches. When a flat workpiece is heated with a simple torch the axis OZ of which is normal to the
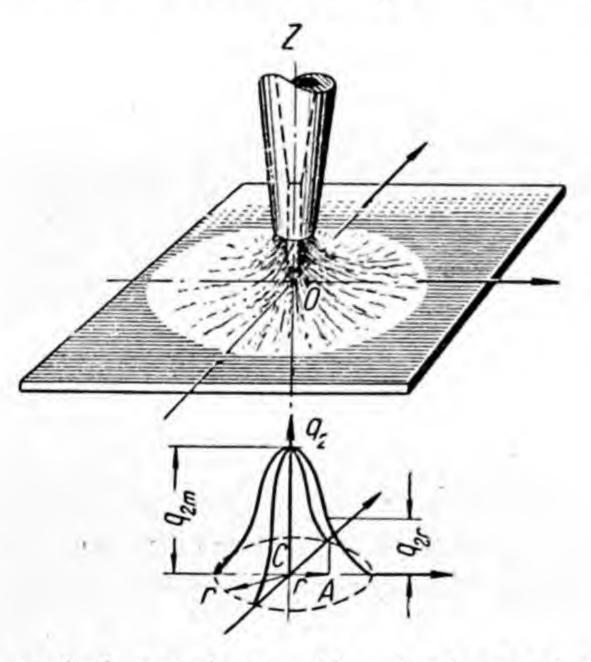
Fig. 53. Distribution of specific heat flux, q_2, over the hot spot with a radius r at 90° flame angle for a simple torch

metal surface, the hot spot is practically a circle whose centre coincides with the axis of the flame (Fig. 53). The specific heat flux in this case is distributed over the area of the circle rather irregularly, but symmetrically relative to the flame axis, being the densest at the centre and the lowest on the periphery, as the temperature and velocity of the flame fall in the same direction.

An approximate idea of how the specific heat flux is distributed across the hot spot can be obtained from the temperature gradient in thin sheets of metal heated with a quickly moving or intermittent flame.

Temperature measurements have shown that it is at its highest at the centre of the spot and almost uniform throughout

the thickness of the metal. Towards the periphery, the temperature decreases and differs greatly between the top and bottom of the specimen (Fig. 54).
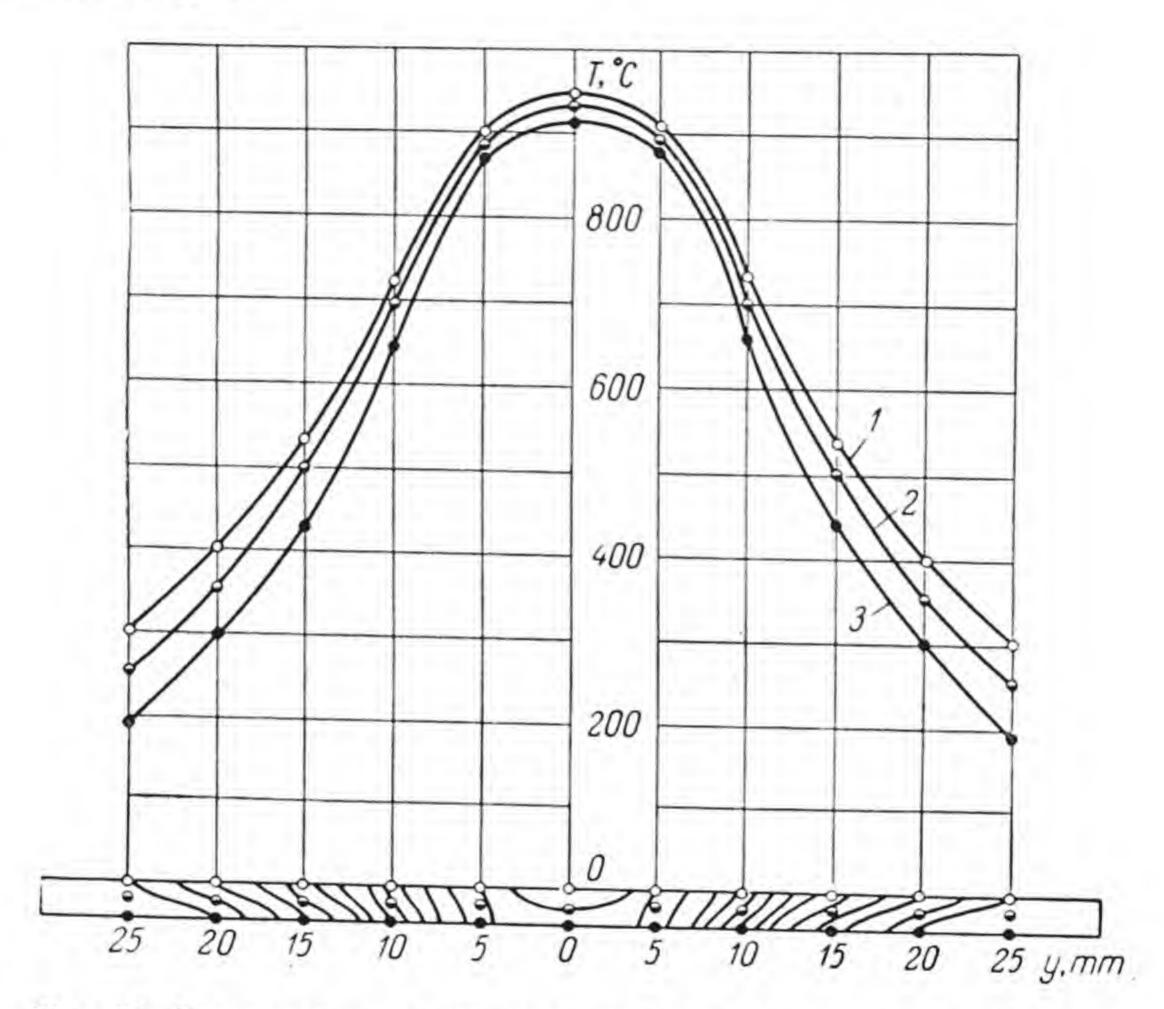
Fig. 54. Temperature gradient in sheet metal across the direction of flame travel when the temperature along the axis is at its highest. Experimental conditions: specimen, steel ЭЯ1Т, 2 mm thick; tip size No. 6; flame angle $\varphi = 90°$; rate of torch travel $\overline{\omega} = 80$ mm/min; effective heat input, $q = 1,100$ cal/sec; oxygen-to-acetylene ratio, $\beta_0 = 1.0$;

1 — top of specimen; *2* — mid-section; *3* — bottom of specimen

The temperature in the middle of the specimen is distributed in a direction normal to the flame and obeys a law which is very close to the normal (Gaussian) distribution law. As the specific heat flux is approximately proportional to the instantaneous temperatures of the hot spot, the flame of a simple torch normal to the metal surface may be regarded as a normal circular source of heat with the specific heat flux distributed normally over the area of the circle (Fig. 53):

$$q_2(r) = q_{2m}e^{-kr^2},$$

$$(VIII.2)$$

where: $q_2(r)$ = specific heat flux at point A of the hot spot, cal/cm²-sec;

q_{2m} = maximum specific heat flux at point O, cal/cm²-sec;

e = base of natural logarithms, 2.718;

r = radial distance from point A to the flame axis, cm;

k = concentration coefficient describing the form of the normal distribution curve, 1/cm².

The maximum specific heat flux is proportional to the total heat input, q cal/sec, from a normal circular source of heat (or to the effective heat input, see Sec. 4):

$$q_{2n} = kq\,\pi, \qquad\qquad \text{(VIII.3)}$$

Table 11 gives the specific heat flux concentrations for tip sizes Nos. 1 through 7 of a simple torch according to N. N. Rykalin and M. Kh. Shorshorov. The measurements were made for an oxygen-to-acetylene ratio $\beta_0 = 1 : 1$, the flame being at right angles to the metal surface ($\varphi = 90°$), and the tip-to-metal distance $h = 5/4L$ (lenght of the inner cone). The steel specimen was 6 mm thick and was traversed by the flame at the rate $\omega = 500$ mm/min.

With greater tip sizes the maximum heat flux q_{2m}, of simple torches increases (Table 11), but other than in proportion to the fuel-gas flow. At the same time, the concentration coefficient, k, decreases, and the hot spot grows in size (Fig. 55).

The distribution of the heat flux over the hot spot in the case of a simple torch is above all governed by the flame angle, the average velocity of the mixture and the tip-to-work distance.

Table 11

Thermal Properties of the Oxy-acetylene Welding Flame
(Simple Torches)

Tip size	Orifice diameter, mm	Acetylene flow, llt per hr	Inner çone length, mm	Effective heat Input, q, cal/sec	Efficiency, per cent	Concentration coefficient, k, 1/cm	Max. specif c heat flux, q_{2m}, cal/cm²-sec
1	1.0	150	9	380	72	0.39	47
2	1.3	250	10	600	68	0.35	67
3	1.6	400	11	720	51	0.31	72
4	2.0	600	12	920	44	0.28	82
5	2.5	1,000	14	1,270	36	0.23	93
6	3.0	1,700	15	1,750	29	0.20	111
7	3.5	2,600	17	2,250	25	0.17	122

In the case of a flat work and an inclined flame, the hot spot is other than circular, and the specific heat flux is distributed over its area asymmetrically relative to the point O in any direction except the axis OY (Fig. 56).

As the flame angle, φ, decreases, the streamer ahead of the inner cone grows in length, and the flame spread over the metal
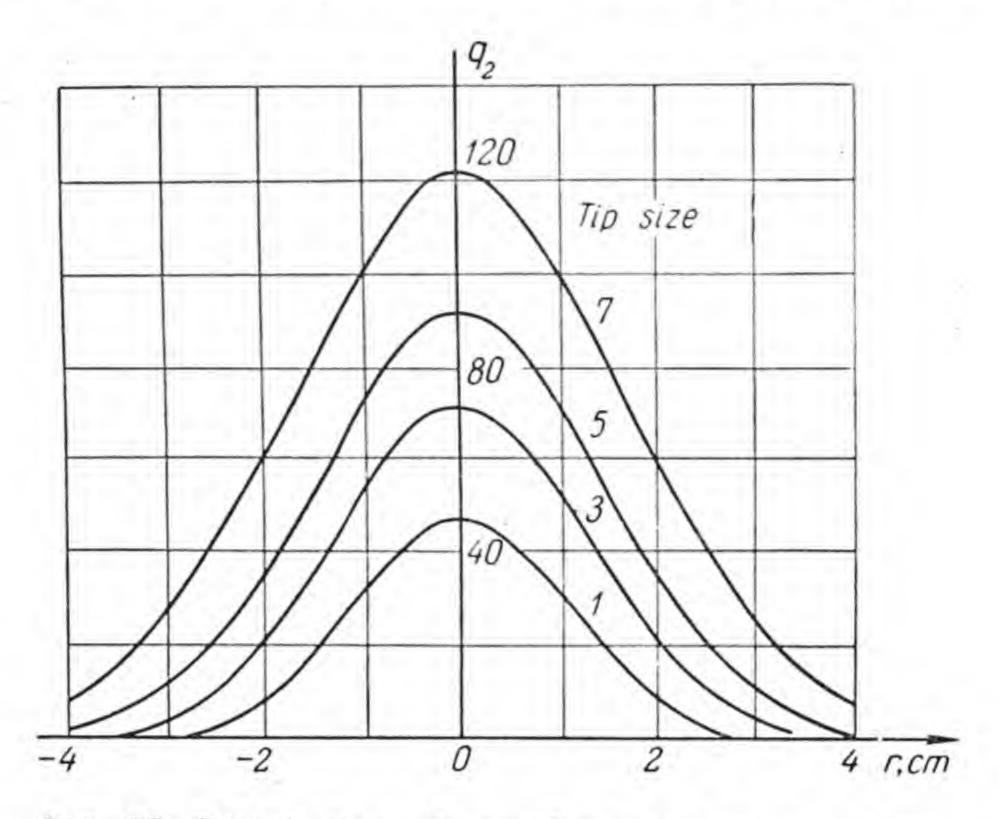
Fig. 55. Distribution of specific heat flux, q_2, across the hot spot for an elementary flame and tip sizes 1, 3, 5 and 7

surface decreases. The hot spot becomes elongated and is shifted towards the inner cone, while the spot width somewhat decreases.

Measurements on thin specimens heated with a stationary inclined flame for a short interval of time have shown that as the flame angle is decreased, the specific heat flux along the projection OX of the flame axis increases beneath and forward of the inner cone and decreases behind it (Fig. 57a). In addition, the heat flux is concentrated along the axis OX (Figs 56 and 57b). The increase in the specific heat flux beneath and forward of the inner cone with decreasing flame angle is due to the increased velocity of the hot gases near the metal surface.

As the tip-to-work distance, h, is decreased, the specific heat flux increases in the centre of the spot and decreases towards its periphery, concentrating still more along the axis of the flame.
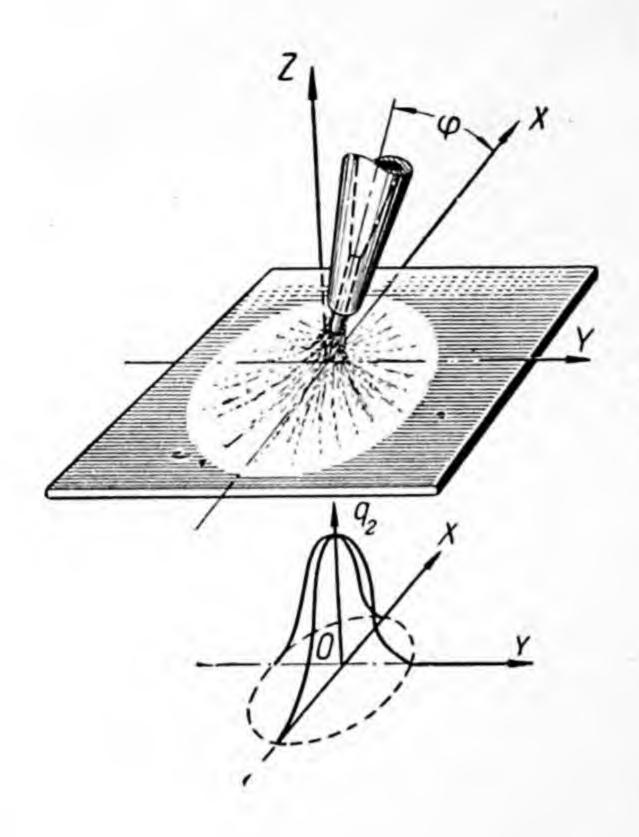
Fig. 56. Distribution of specific heat flux, q_2, across the hot spot for an elementary flame at other than 90° flame angle

(a)   (b)
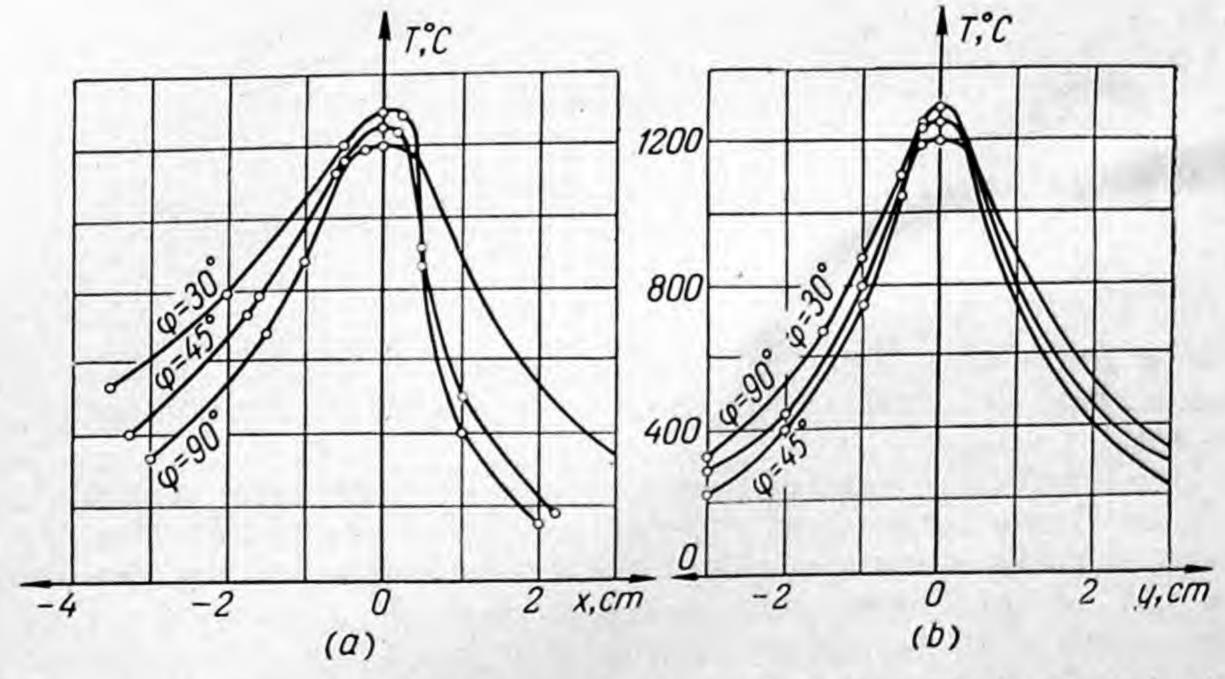
Fig. 57. Temperature gradient in sheet metal at the end of a short heating cycle with a stationary elementary flame as a function of flame angle φ. Experimental conditions: steel ЭЯ1Т, 1.5 mm thick; tip size No. 3; oxygen-to-acetylene ratio $\beta_0 = 1.15$; $L = 10.5$ mm; $h = \frac{5}{4} L = 13$ mm; heating time 3 sec;

(a) along axis OX (projection of the flame on the sheet); (b) along axis OY (perpendicular to axis OX) (see Fig. 56)

Indeed, as the cone and the hottest intermediate zone around it approach the metal surface, the temperature and velocity of the hot gases striking at the central part of the hot spot rise. The hot spot, however, shrinks in size because the hot gases now turn away from the metal surface nearer to its centre. When the distance h is less than the length of the inner cone, L, the latter is deformed as it strikes the metal. The ignition zone and the hottest intermediate zone together with the maximum specific heat flux are shifted away from the spot centre and are now located around it.

Where the average velocity w of the mixture is increased (or the orifice diameter is reduced while the velocity is left unchanged), the velocity of the hot gases and the specific heat flux in the central area of the hot spot increase.

(2) The Flame of Composite Torches. Composite torches are convenient in that the individual flames can be arranged so as to obtain the desired shape and size of the aggregate flame and to distribute the heat flux across the workpiece at will.

Composite torches may be of either the multiflame or of the slot type. In a multiflame torch, the flames may be arranged in a line or in rows. Where the flames are spaced closely apart, the individual flames merge into a single one some distance from the orifices. (Fig. 58). In the slot type, there is a single annular orifice; it may be of any desired outline and will produce a single flame of the same shape as the slot (Fig. 59a). Depending on the shape of the workpiece or the purpose of heating, composite torches may have flat (Figs 58a and 59a, b), circular (Fig. 58b) or odd-shaped heads (Fig. 58c). Multiflame torches are the more commonly used type.

Fig. 60 shows the instantaneous temperature gradients across the hot spot on a specimen heated for a short time with an annular multiflame torch employed for preheating the metal in oxygen-cutting (Fig. 59). The highest temperature and, consequently, the maximum specific heat flux are some distance from, rather than in, the centre of the hot spot. This distance is the radius of the circle round which the flames are arranged. The heat flux is more concentrated close to the axis of the flame and its distribution approaches the normal one (Eq. VIII.2) with decreasing radius of this circle. Given the same fuel-gas flow, the maximum specific heat flux and heating efficiency will be obtained with a greater number of flames and with a smaller radius of the flame circle.

A straight or in-line multiflame torch head has one row of flames arranged in a line at a pitch f (Fig. 58a). The specific heat

flux of this torch is distributed across the hot spot nonuniformly (Fig. 61), being greater beneath the inner cones and smaller
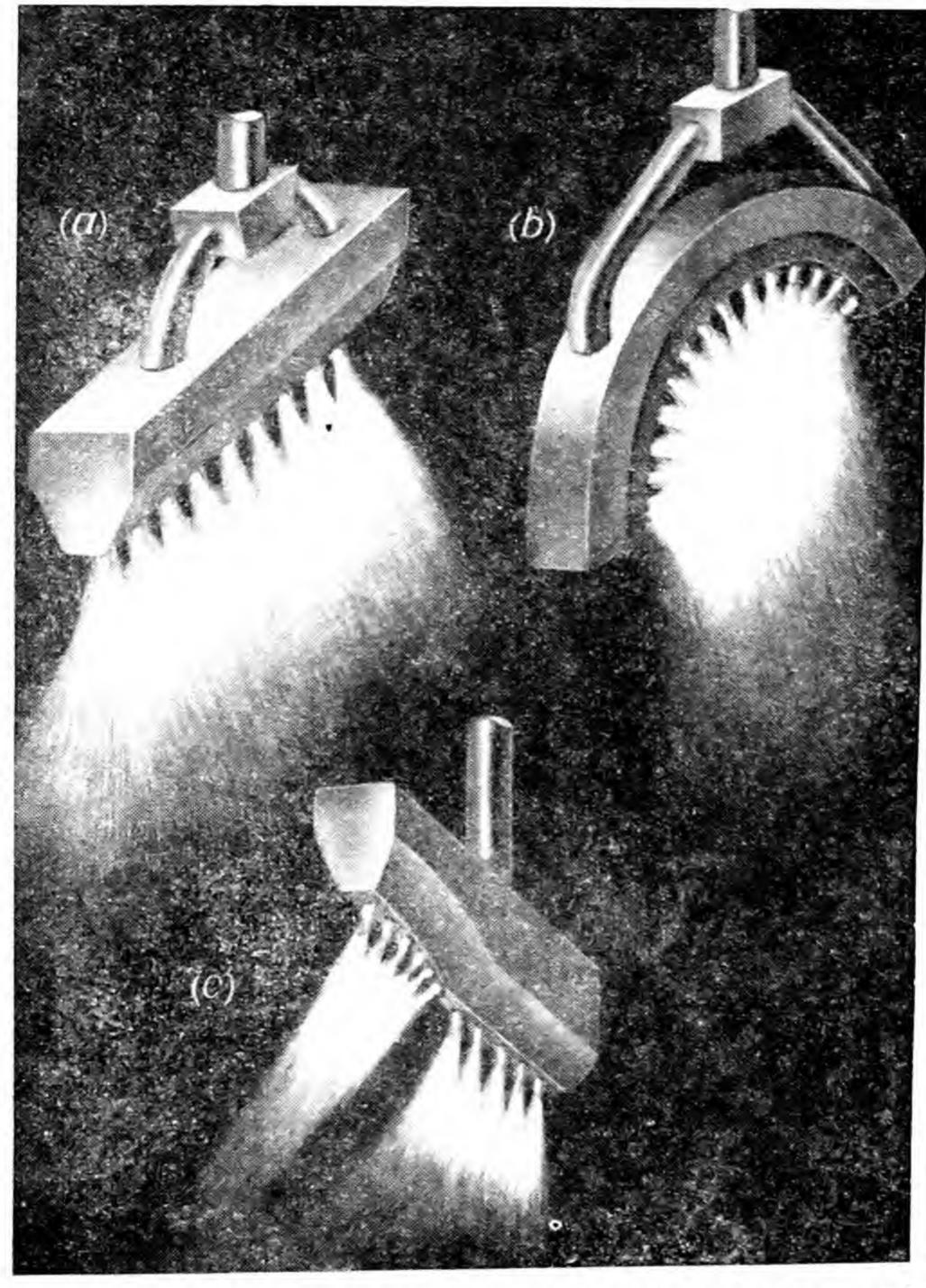
Fig. 58. Single-row multiflame torches for welding and flame hardening:
a — in-line torch (with flames on a flat surface); *b* — ring torch (with flames on a circular surface); *c* — odd-shaped torch

midway between the flames. In the case of straight torch heads used for flame hardening (the torch head travels along the axis *Y*, Fig. 61) and for the automatic welding of longitudinal seams

on thin-walled pipes (the torch travels along the axis X), the nonuniformity usually does not exceed 10 to 15 per cent. This nonuniformity is less for heads with closer spaced flames and for the greater tip-to-metal distance
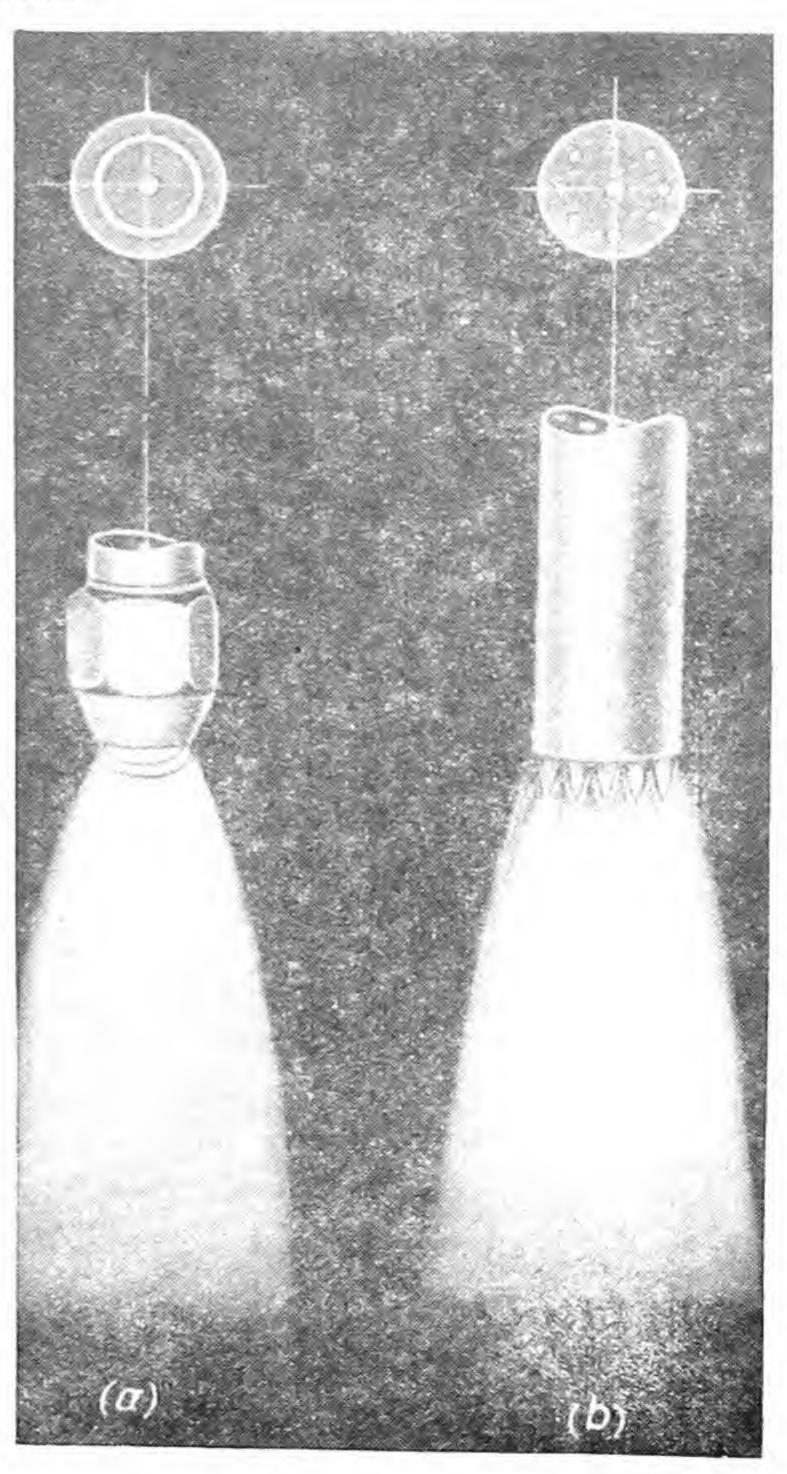
Fig. 59. Annular cutting torches:
a — slot type; *b* — multi-flame type

The distribution of the specific heat flux along *OY* within the effective length of the torch head is close to the normal one. The concentration of the heat flux in the axial plane of the torch head increases with decreasing diameters of the individual orifices and fuel-gas flow and with increasing flame spacing. When the condi-

tions are reversed, the flames grow in size. At the same time, owing to the fact that the hot gases are concentrated along *OX*, the flame spreads more over the work along the axis *OY*, and the heat flux becomes less concentrated in the axial plane of the torch head.

Multiflame multirow torch heads distribute the heat flux over the metal surface more uniformly than do singlerow multiflame heads. Such heads are emp- loyed in flame hardening and, sometimes, in gas-pres- sure welding. In the former application, the metal can be heated with both a stationa- ry and a progressing flame. In the latter application,
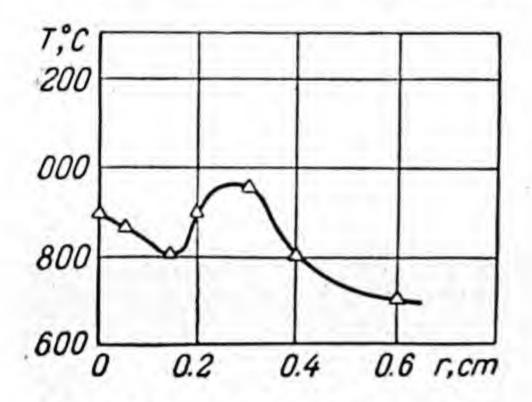
Fig. 60. Temperature gra- dient across the hot spot in low-carbon steel sheet 5 mm thick in the case of an annular multiflame torch employed for preheating in cutting (the flames are ar- ranged all the way round a circle 6 mm in diameter)
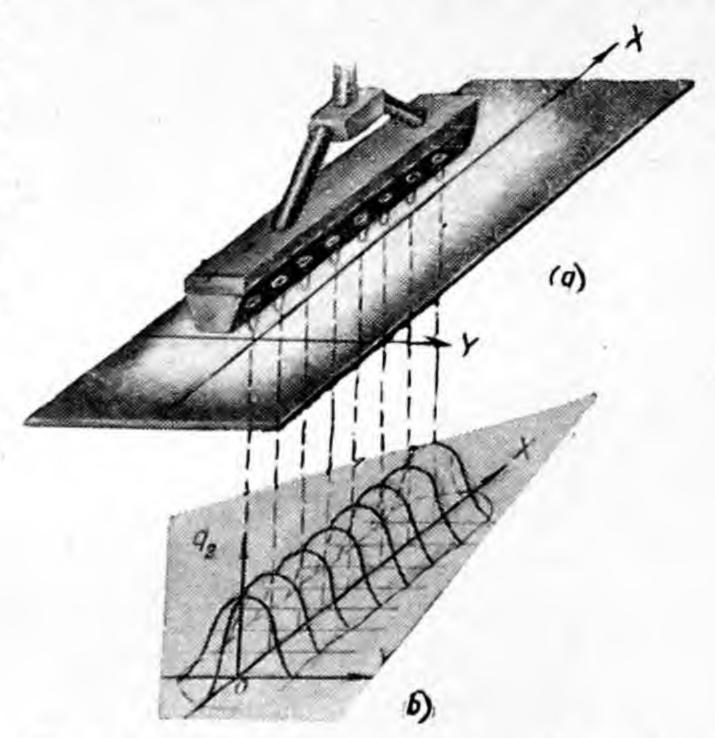
Fig. 61. Distribution of **specific** heat flux across the hot spot in the case of an in-line torch:
a — in-line multiflame torch; *b* — actual distribution of specific heat flux

the work is only heated at the fusion faces with a stationary flame.

In the case of a flat multirow multiflame torch head, the spe- cific heat flux within the effective area of the head is distributed across the hot spot more or less uniformly. Outside the effective area, i. e., across the hot spot washed by the aggregate flame envelope, the specific heat flux falls progressively towards the periphery of the flame envelope (Fig. 62). The heat flux is more concentrated where the flames are spaced closer and the acetylene flow per unit working area of the head is greater. By way of example, the distribution of the heat flux across the ends of steel

bars heated for flame hardening or gas-pressure welding will be acceptably uniform with the jets spaced 4 to 6 mm apart and with an acetylene flow of 3 to 4 litres per hour per sq mm of area.
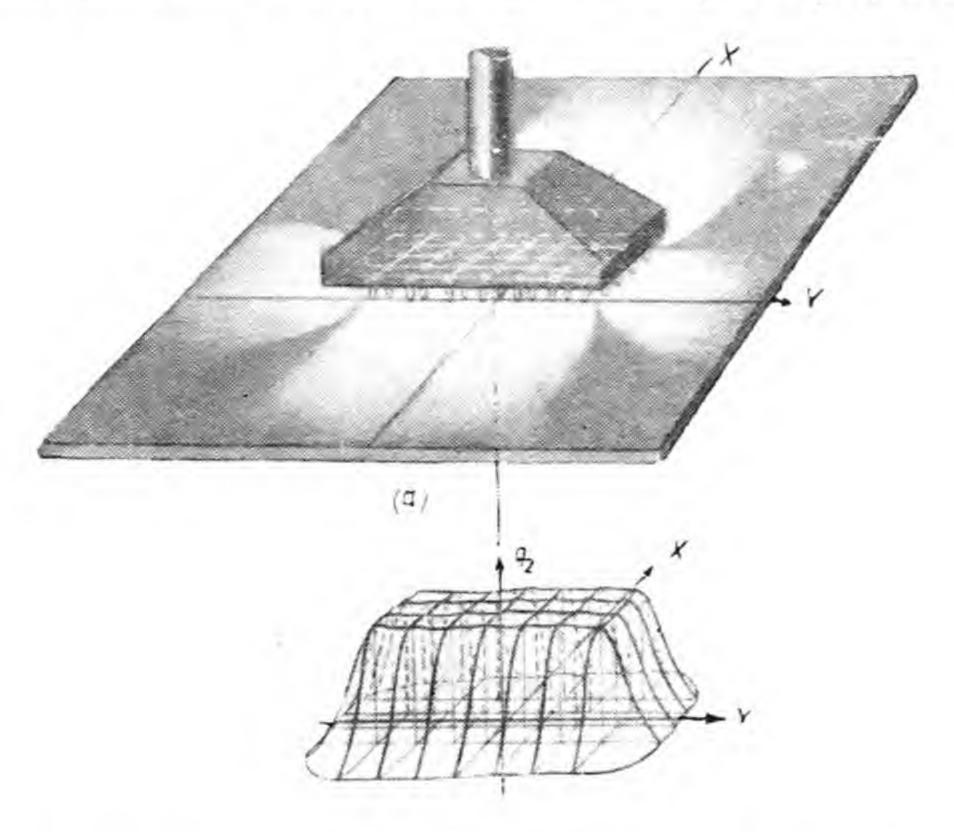
Fig. 62. Distribution of specific heat flux across the hot spot in the case of a multirow torch:
a — sketch of a multirow torch; *b* — actual distribution of specific heat flux

4. Effective Heat Input. The Efficiency of Welding and Heating

There is much difference between heating the metal with a stationary and a moving flame.

In the case of a stationary flame, the heat content, $Q(t)$ cal, of the work varies nonlinearly with the time of heating, t sec (Fig. 63a). At the beginning, when the metal surface in contact with the flame is cold, the heat content increases at a higher rate. As the temperature of the metal surface rises, the heat content of the work grows increasingly more slowly, tending to some steady-state limit Q_{lim}. This limit characterises the steady-state condition under which the temperature in any point of the body no longer rises. The heat content of a workpiece can be conveniently measured with a calorimeter.

Let the quantity of heat imparted to the metal by a flame in unit time be called the effective heat input, $q(t)$. For short-

duration heating, the loss of heat by the work to the surroundings may be neglected, and the effective heat input may be expressed as follows:

$$q(t) = \frac{dQ(t)}{dt}.$$

In heating, the effective heat input decreases from its original maximum value q_0 nonlinearly with the time of heating and tends to zero. This nonlinear change of the heat content of the work and of the effective heat input is due to the nature of the heat exchange between the flame and the metal.
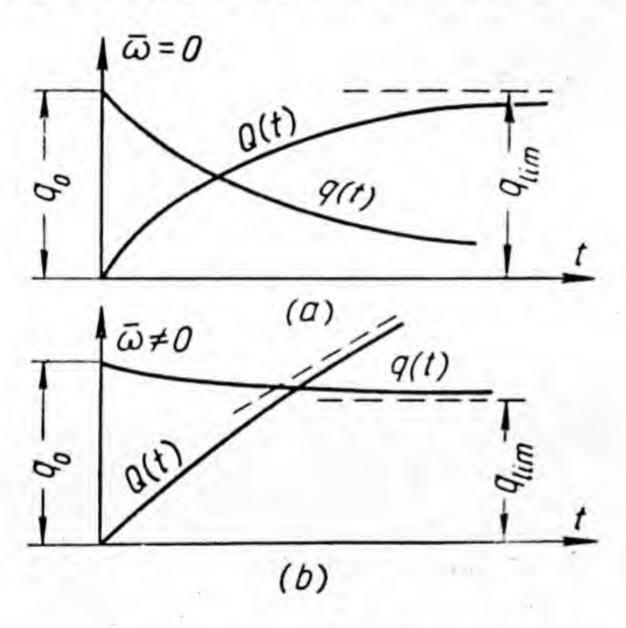
Fig. 63. Changes in the heat content, $Q(t)$, of a plate and in effective heat input, $q(t)$, in the case of (a) a stationary flame and (b) an advancing flame

More specifically, the rate of convective and radiant heat transfer is reduced as the metal is raised in temperature and the temperature difference between the flame and the metal surface is decreased.

In practice, the steady state can be attained with thin tungsten sheets, but it is unattainable for sheets of steel and nonferrous metals having a relatively low melting point.

Where the work is heated with a flame moving at a uniform rate and in a straight line, the heat content $Q(t)$ increases nonlinearly with the time of heating, but more uniformly than with a stationary flame (Fig. 63b). A plausible explanation may be that the moving flame is continually coming in contact with ever colder areas of the work. In steady-state heating, the temperature field moving with the flame remains constant in size and so does the rate of heat exchange at any given point of the travelling field. For this reason, the heat content becomes a linear function of time, and the effective heat input is constant (Fig. 63b).

The steady state in heating with a moving flame is attained the sooner, the higher is the rate of flame travel. Within the limits
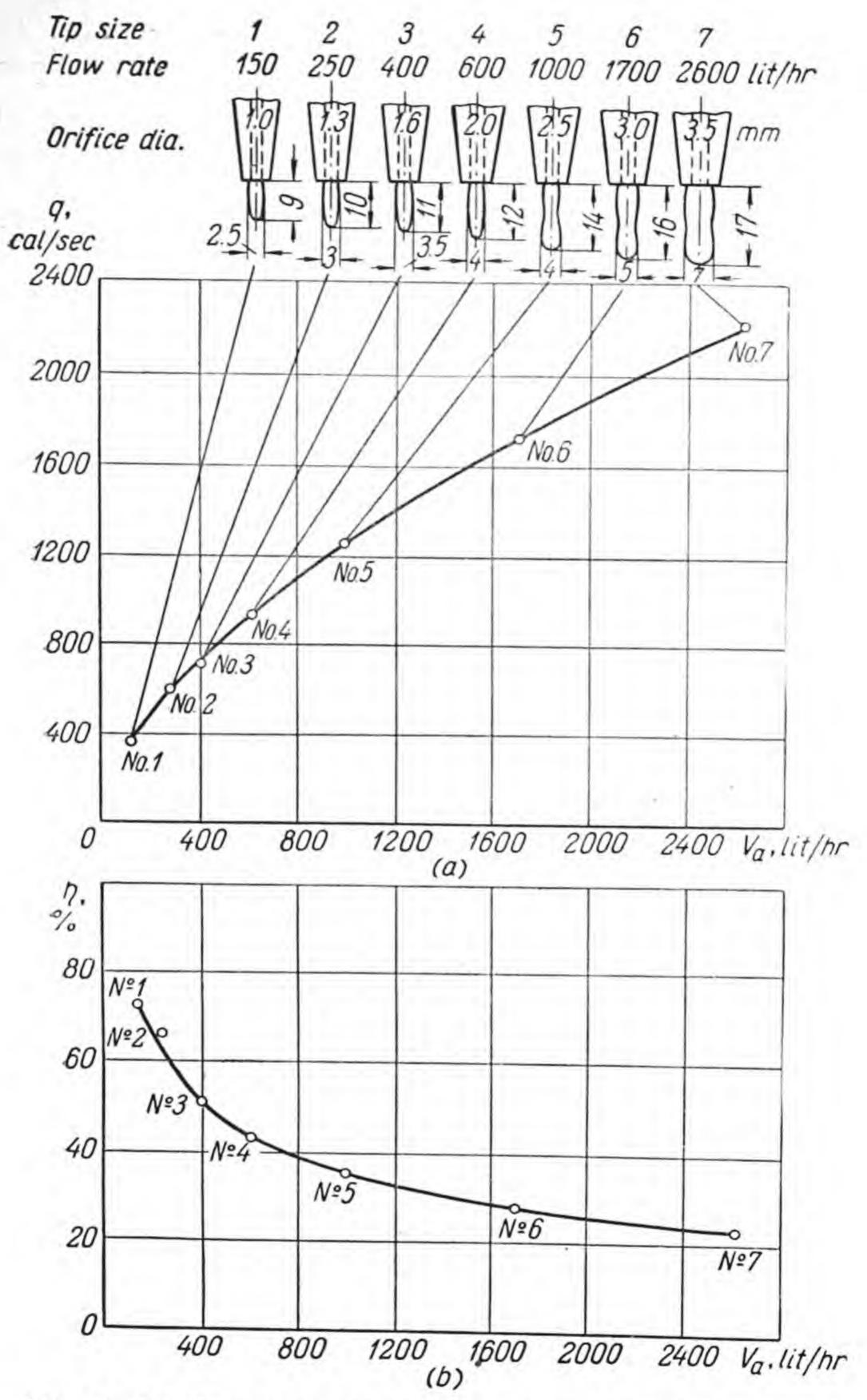
Fig. 64. (a) Effective heat input and inner cone length and (b) heating efficiency plotted against the rate of acetylene flow (tip size). Experimental conditions: $\varphi = 90°$; $h = \frac{5}{4} L$; oxygen-to-acetylene ratio $\beta_0 = 1$; the average velocity of gas w, from 108 to 152 m/sec; rate of torch travel $\overline{\omega} = 500$ mm/min; specimens, low-carbon steel $110 \times 50 \times 6$ mm

10*

of flame travel employed for the welding and cutting of steel, it takes one but a few seconds to attain the steady-state conditions. As the rate of flame travel increases, the difference between the initial and final or limiting values of the effective heat input, q_0 and q_{lim}, is also reduced.

For the purpose of our discussion the final value, q_{lim} cal/sec, for a moving flame will be further referred to simply as the
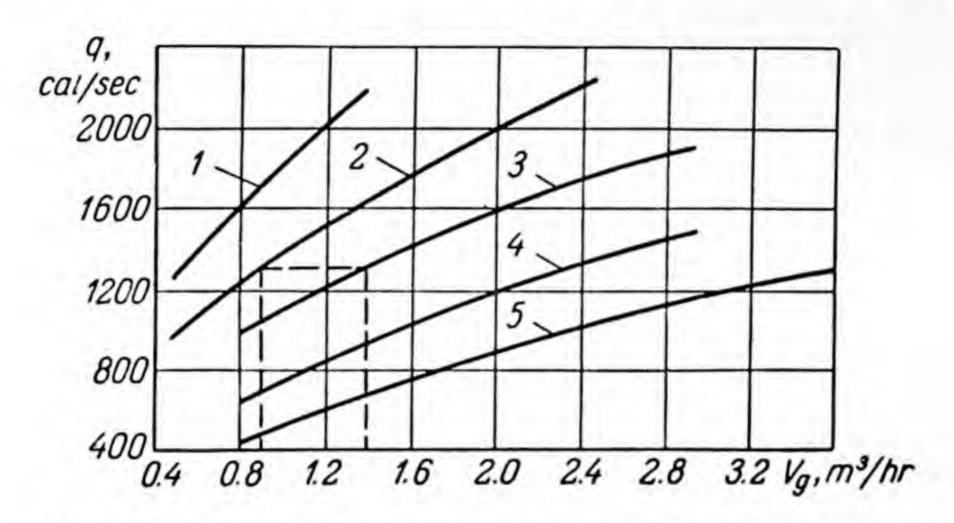
Fig. 65. Effective heat input for various oxy-gas mixtures burned in annular multiflame torches (see Fig. 59*b*) related to rate of fuel flow. Experimental conditions: $\varphi = 90°$; $h = \frac{5}{4} L$; rate of torch travel $\bar{\omega} = 900$ mm/min; specimens, $110 \times 50 \times 5$ mm; oxygen-to-fuel gas ratios; *1* — $\beta_0 = 3.5$ for propane-butane; *2* — $\beta_0 = 1.15$ for acetylene; *3* — $\beta_0 = 1.5$ for methane; *4* — $\beta_0 = 0.8$ for coke-oven gas; *5* — $\beta_0 = 0.4$ for hydrogen

effective heat input and designated by the letter q without any subscript.

The effective heat input is mainly dependent on the fuel-gas flow, increasing with it, though other than linearly (Figs 64*a* and 65).

For example, in the case of a single-flame oxy-acetylene torch, the increase in the acetylene flow rate V_a from 150 to 2,600 lit/hr (i. e., from tip size No. 1 to No. 7) raises the effective heat input q from 350 to 2,200 cal/sec (Fig. 65).

The relative efficiency of the oxy-acetylene flame, η_r, is the ratio of the effective heat input, q, to the total flame output, q_L, for the low heat value of the fuel:

$$\eta_r = \frac{q}{q_L}. \qquad (VIII.4)$$

For acetylene, the low heat value is 12,600 cal/lit (at 20° C and 760 mm Hg), and the relative efficiency of heating with the oxy-acetylene flame will be:

$$\eta_r = \frac{q}{\frac{12{,}600}{3{,}600} V_a} = \frac{q}{3.5 V_a},$$

(VIII.5)

where V_a is the volume rate of acetylene flow in litres per hour.

The increase in the tip size of a single-flame torch from No. 1 to No. 7 (or, which is the same, in the acetylene flow) cuts down the efficiency of steady-state heating from 72 to 25 per cent (Fig. 64b). Given the same thickness, the thermal efficiency * and the efficiency of welding or heating rise, however, owing to the increased effective heat input and specific heat flux, q_2.

The decrease in the relative efficiency of heating, η_r, with an increase in the fuel-gas flow is due to several factors. For one thing, the larger tip sizes and, consequently, the higher fuel-gas flows are less efficient in such a case, because the flame exchanges heat with a hotter metal surface. For another, the flame grows in size, the supply of air is handicapped, the products of incomplete combustion have to burn in the distant areas of the flame envelope, and the heat flux is spread still more. Some of the heat is also lost to the waste gases. As will be seen, other variables of heating, as well as the size of the work and the thermal properties of the metal also have a bearing on the effective heat input, though to a lesser degree than the volume rate of fuel flow.

(1) The Flame Angle. In welding and heating, the operator holds the torch in his right hand so that the flame makes an angle with the metal surface, as shown diagrammatically in Fig. 66. The flame can be moved either to the left or to the right. Measuring the flame angle, φ, from the tip towards the metal surface in the direction of welding (or heating) gives $\varphi > 90°$ for the leftward technique and $\varphi < 90°$ for the rightward technique.

The effective heat input is the greatest with the rightward technique and with a flame angle of 60° (see Fig. 66). As the flame angle increases from 60 to 160°, the effective heat input is almost halved. The effective heat input is greater with the rightward than with the leftward technique due to the fact that the hottest intermediate zone of the flame moves across a colder metal in the former case and across the metal preheated by the streamer in the latter.

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* In welding or heating, by thermal efficiency should be meant the quantity of heat raising the temperature to the desired point within the desired area.

Still, although less heat per unit time is imparted to the metal with the leftward technique, the available heat is utilised better, the reason being as follows. In rightward heating, the hottest intermediate zone is the first to come in contact with the metal, and the temperature of the metal quickly rises to the maximum. The colder zones of the flame following in its wake cannot raise the temperature any more, though they slow down cooling. With the leftward technique, the metal is first heated by the streamer and then by the intermediate zone; low in the beginning, the rate of heat transfer steadily increases. Preheated with the streamer, the metal is then raised by the intermediate zone to a higher
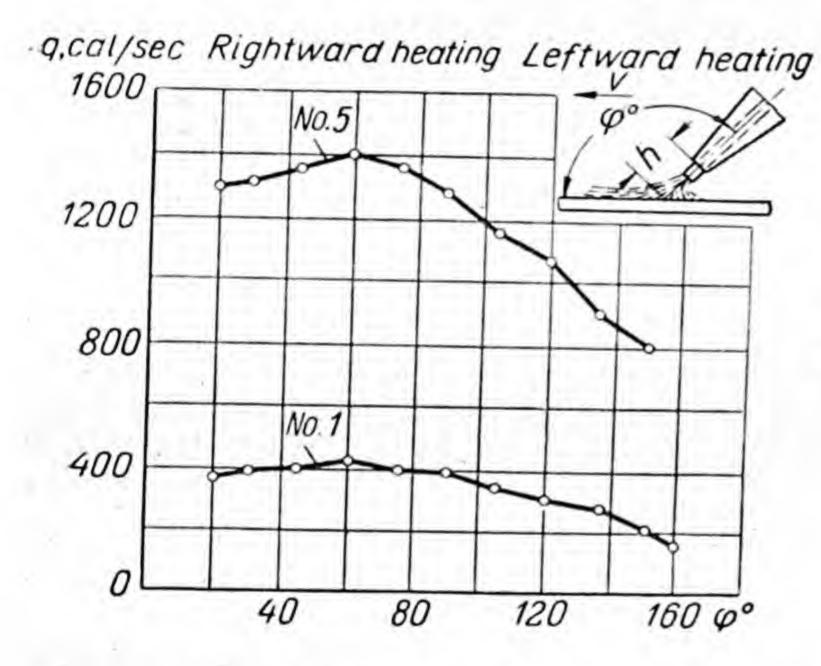
Fig. 66. Effective heat input plotted against flame angle, φ, for tip sizes 1 and 5. Experimental conditions: $h = L$; other factors same as in Fig. 64

temperature than in rightward heating. Putting it another way, despite the lower relative efficiency of heating, the thermal efficiency is higher with the leftward technique.

The highest rates of heating are obtained with the leftward technique, or with a flame angle of more than 90°, but the angle should be progressively decreased for the heavier gauges and the greater thermal conductivities.

Experience has shown that the leftward technique ensures higher rates in the manual welding of square butt welds in light-gauge (1-3 mm) sheet. In light gauges, the inner cone of the flame is practically the same distance from the surface of the workpiece in both rightward and leftward welding, while the latter offers the same advantages as in ordinary heating.
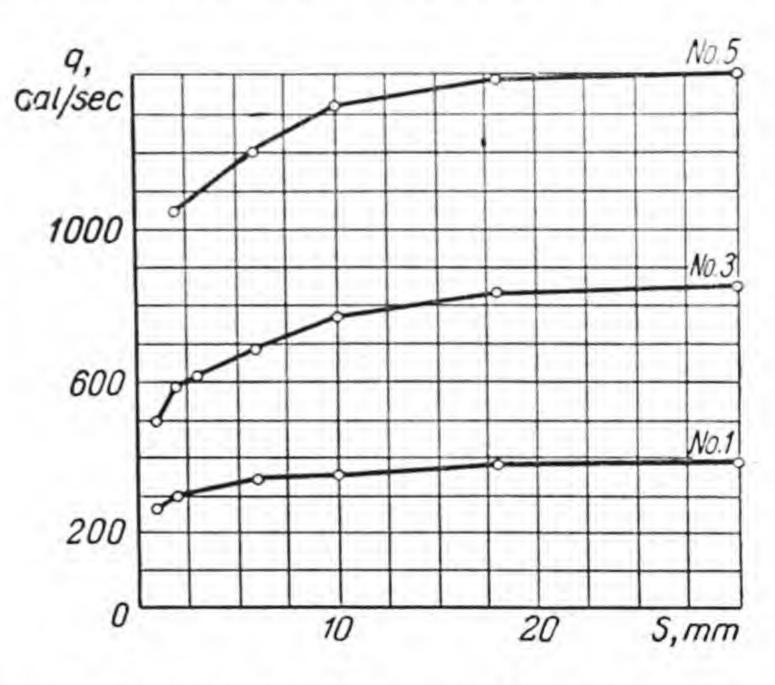
The same applies to the high-speed automatic welding of longitudinal seams on thin-walled pipes with straight multiflame heads in which the jets are inclined at 110°, and to the flame hardening of steel.

When it comes to the manual welding of plate more than 5 mm thick, especially where the edges of the work have to be veed, bevelled or otherwise prepared, the rightward technique proves better, because the hottest intermediate zone reaches deeper into the groove than in leftward welding, the heat flux is more concentrated, the rate of heat transfer increases, and a narrower heat-affected zone is produced. This difference becomes more pronounced on the heavier gauges.

(2) Metal Thickness. As the thickness of the metal increases, the effective heat input also increases, especially (by 30 to 40

Fig. 67. Effective heat input for tip sizes 1, 3 and 5 as a function of metal thickness *S*. Experimental conditions same as in Fig. 64.

per cent) in the thickness range from 1 to 15 mm (Fig. 67). On thicknesses from 15 to 30 mm this increase is insignificant (2 or 3 per cent).

The increase in the effective heat input is due to the fact that in a thicker metal more heat is abstracted into the body of the metal, and the temperature of the metal surface falls. The welding (or heating) rate is, however, reduced because less heat is imparted to unit thickness of metal per unit time (q/s).

(3) The Average Velocity of the Mixture. The effective heat input may be raised by as much as 30 per cent by increasing the average velocity of the mixture, w, from 60 to 170 m/sec or by reducing the orifice diameter, d, from 2.2 to 1.3 mm while keeping the acetylene flow constant at 400 lit/hr (Fig. 68). Any further increase in the rate of fusion welding through a higher velocity of the mixture is impossible as the molten metal will be forced

out of the pool. As to other flame metal-working processes, increasing the velocity w is a very efficient way of raising both the relative and thermal efficiencies and the heating rate.
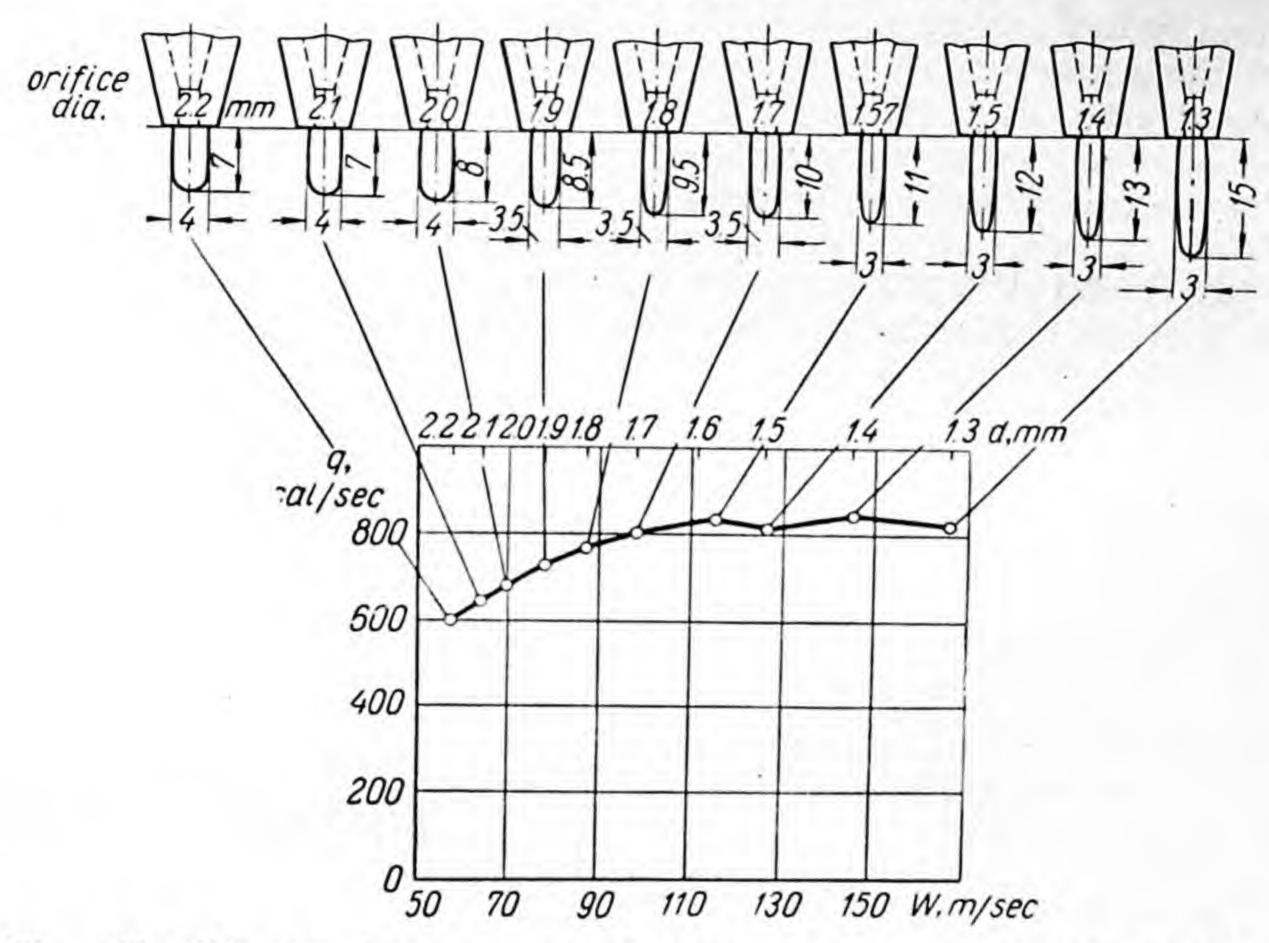
Fig. 68. Effective heat input and inner cone length plotted against average velocity of mixture, w, or orifice diameter, d, at a constant rate of acetylene flow of 400 lit/hr (tip size No. 3). Experimental conditions: $h = L$; other factors same as in Fig. 64

(4) The Oxygen-to-Gas Ratio. As the proportion of oxygen in the mixture increases to some limit, the flame velocity rises (Fig. 69), the inner cone and streamer contract, and more acetylene is burned near the cone. At the same time, the intensity of combustion is stepped up, the heat is more concentrated, less heat is lost to the waste gases, and the effective heat input is increased (Fig. 70a).

The highest effective heat input is usually obtained with the oxygen-to-acetylene ratio $\beta_0 = 2.2$ or 2.4 and is 20 to 25 per cent greater than with the normal flame for which $\beta_0 = 1.1$-1.2 (Fig. 70a).

When the proportion of oxygen in the mixture is in excess of its combining proportion with the acetylene ($\beta_T = 2.5$), the excess oxygen, instead of supporting the combustion, becomes a liability, and the flame velocity falls (see Fig. 69), while the inner cone

and streamer grow in size. The spread of the flame and the increased losses of heat to waste gases and to heating the excess oxygen bring down the effective heat input (Fig. 70a).
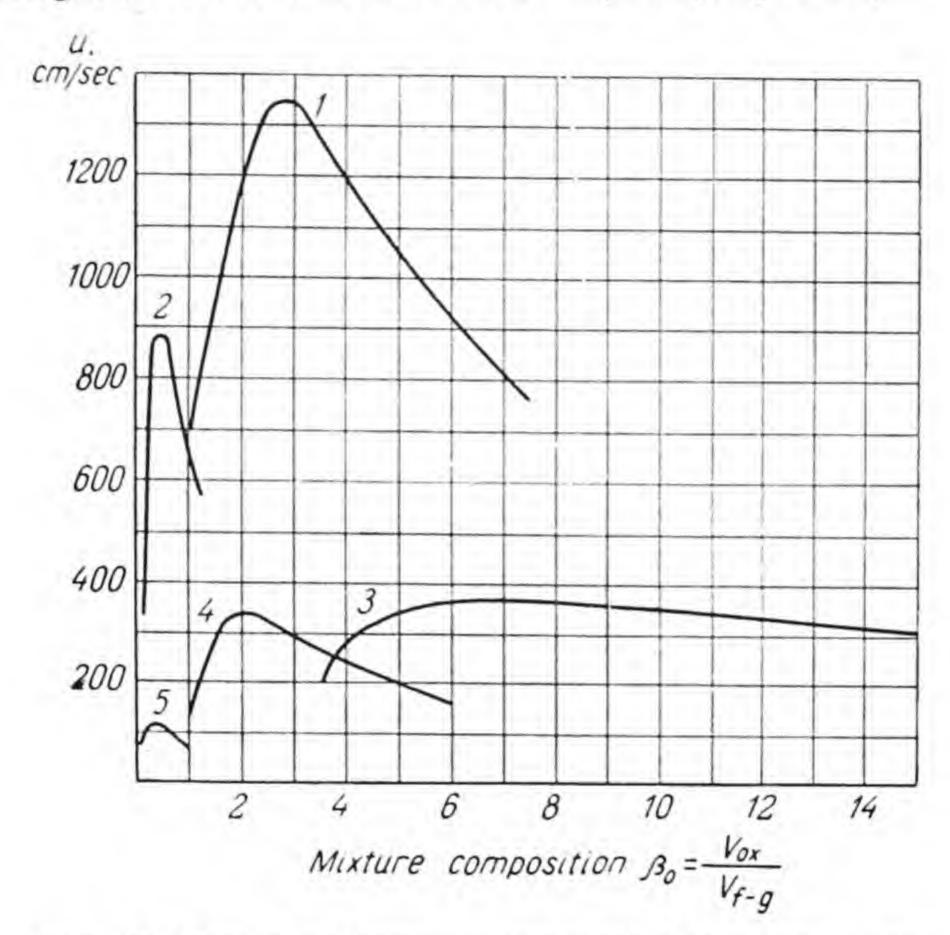
Fig. 69. Flame velocity, u (cm/sec), for various oxygas mixtures as a function of oxygen-to-fuel gas ratio (by volume) $\beta_0 = \dfrac{V_{ox}}{V_{f-g}}$:
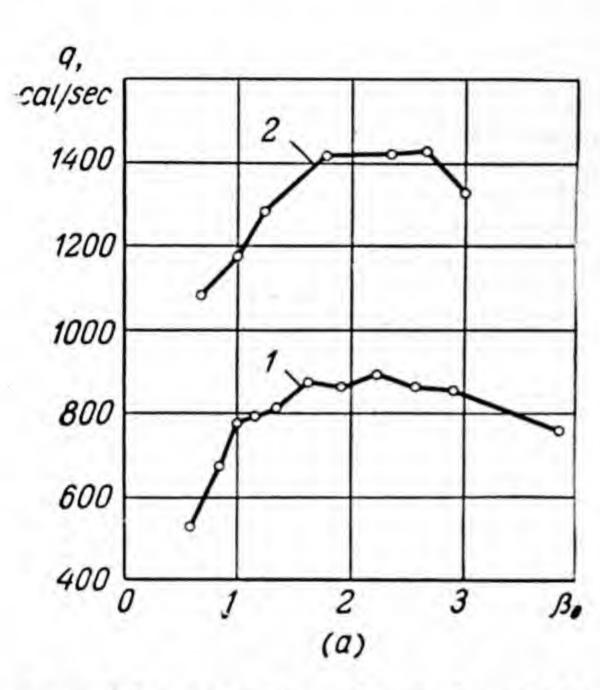
1 — for acetylene; *2* — for hydrogen; *3* — for propane; *4* — for methane; *5* — for carbon dioxide

Fig. 70b relates the effective heat input of other oxy-gas mixtures to the oxygen-to-gas ratio β_0. The values of β_0 with which the highest effective heat input can be obtained are given below:

Acetylene	2.3
Methane (or natural gas)	2.0
Coke-oven gas	0.8
Hydrogen	0.4
Propane-butane mixture	3.5

These values of β_0 are always lower (by about 10 per cent) than the combining proportions, β_T. This is because some of the gas burns with the oxygen of the air which is drawn in by the various flame zones. With these values of β_0, both the relative

and thermal efficiencies * of heating rise, and so does the heating rate, but the flame becomes oxidising. Therefore, a flame with these values of β_0 may only be employed where the metal is heated to below the point of intensive oxidation, unless oxidation is
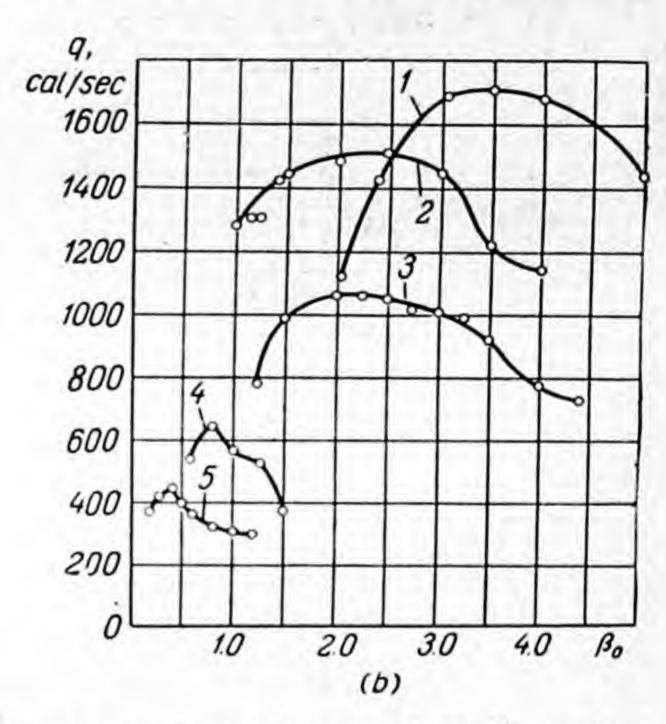
Fig. 70. Effective heat input for various oxy-gas mixtures plotted against oxygen-to-gas ratio, β_0:

a — oxy-acetylene flame, elementary torch, with a rate of acetylene flow $V_a = 400$ (at *1*) and 800 lit/hr (at *2*) at a constant average mixture velocity $w = 115$ and 130 m/sec, respectively. Experimental conditions: $h = L$; tip diameter, d, from 1.4 to 3.5 mm for $V_a = 400$ lit/hr and from 2 to 3 mm for $V_a = 800$ lit/hr. The other factors same as in Fig. 64;

b — various oxy-gas flames, annular multiflame torches, $V_a = 800$ lit/hr. Experimental conditions: $\varphi = 90°$; $h = \frac{4}{5} L$; $\overline{\omega} = 900$ mm/min; specimens $100 \times 50 \times 5$ mm;

1 — propane-butane mixture; *2* — acetylene; *3* — methane; *4* — coke-oven gas 5 — hydrogen

the objective of the process, as in flame cutting. The oxidising flame may be used for high-speed flame cutting, flame hardening, corrective heating and some other applications, but it is out of the question in most welding and brazing applications.

In flame cutting, the operator often has no instruments with which to measure the fuel-gas flow. Instead, he has to rely on the appearance of the flame. For most fuels, the oxygen-to-fuel ratios

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* In this case, thermal efficiency is identical with relative efficiency, because the increased heat input is accompanied by a reduced flame size, an increased heat concentration and a higher flame temperature.

adjusted by such a method are very close to the optimum ones, i. e., at which the effective heat input is the greatest. The exceptions to the rule are acetylene and methane for which the ratios as adjusted by flame apperance are below the optimum values.

The oxygen-to-fuel gas ratios as adjusted by sight are given below:

	β_p
Acetylene	1.15
Methane (or natural gas)	1.5
Coke-oven gas	0.8
Hydrogen	0.4
Propane-butane mixture	3.5
Oil-refinery gas (average)	2.0
Shale gas (average)	0.7

In flame cutting and machining it is well advised to replace acetylene by some cheaper combustible. The acetylene flow in the preheating flame is adjusted depending on the thickness of the metal and the job on hand — cutting or machining (see Part Five). In replacing, the flow of the substitute fuel gas should be chosen so that the same efficiencies and heating rates may be obtained as with acetylene.

This is done on the basis of the equivalence factor, ψ, which is the ratio of the substitute-gas flow, V_s, to that of acetylene, V_a, with which the flame has the same thermal characteristics (q, q_{2m} and k) in both cases:

$$\psi = \frac{V_s}{V_a}. \qquad \text{(VIII.6)}$$

M. Kh. Shorshorov and A. K. Minburg have compared the concentration of the specific heat flux for various preheat flames at a constant effective heat input, q. According to them, when the diameter of the slot (see Fig. 59a) or of the flame circle (see Fig. 59b) is maintained constant, the maximum heat flux, q_{2m}, and its concentration, k, may to a first approximation be also considered constant and practically independent of the kind of fuel gas burned. Therefore, where the preheat flame travels in a straight line and at a uniform rate, acetylene may be replaced by any other combustible gas on the basis of the effective heat input obtained with the practical oxygen-to-gas ratios (β_p). The flow rates for substitute gases and acetylene are related in Fig. 65.

Example: In cutting steel plate 100-200 mm thick, an oxy-acetylene preheat flame is used with an average acetylene flow of 0.9 m³/hr. Find the flow rate for methane required to cut this steel. In Fig. 65, we find that the acetylene flow rate of 0.9 m³/hr corresponds to an effective heat input of about

1,300 cal/sec. The same effective heat input is obtained with methane flowing at a rate of 1.4 m³/hr. Then the equivalence factor will be:

$$\psi = \frac{1.4}{0.9} \approx 1.6.$$

Thus, metals may be cut by other fuel gases at about the same rate as with acetylene, provided the same effective heat input is obtained. It takes, however, 40 to 60 per cent more time to raise the start of a cut to kindling point. When the metal is raised in temperature with a stationary flame, its effective heat input q, the specific heat flux q_2 and the heat transfer coefficient α progressively decrease as the metal becomes hotter (see Fig. 63a). For the lower flame temperatures, the rate of this decrease is higher, and the metal is heated more slowly.

When large plate has to be cut, the time of heating at the beginning of the cut is but a small fraction of the overall cutting time and may therefore be neglected. Then the flow rate for the substitute gas may be adjusted on the basis of the effective heat input for a moving flame, thereby effecting an economy of fuel.

When cutting short plate, as well as in flame machining, the time of heating at the beginning of a kerf accounts for a considerable proportion of the overall time. In flame machining, this is because the rate of heating is higher and the cuts are shorter than in ordinary flame cutting; accordingly, the heat input for the substitute gas should be so chosen as to obtain the same time of heating as with the oxy-acetylene flame. As a rule, the equivalence factor in such cases is 1.5 to 2.5 times greater than for cutting.
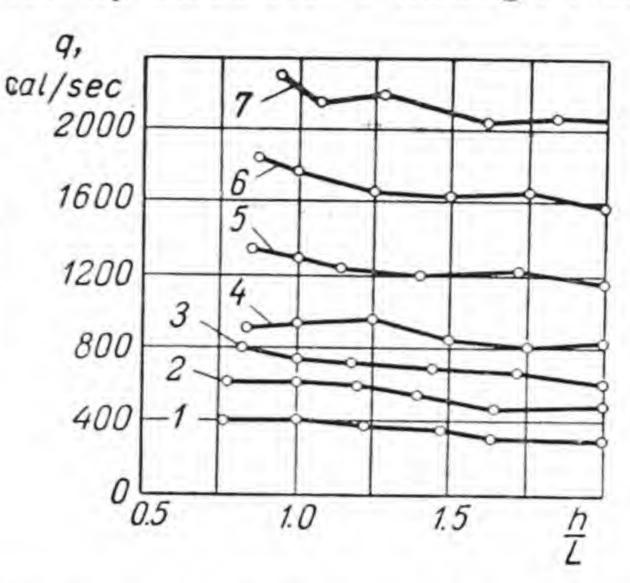
Values of the equivalence factor, ψ, for flame cutting and machining are tabulated in Table 12.

Table 12

Fuel gas	Equivalence factor	
	flame cutting	flame machining
Hydrogen	5.2	—
Natural gas (methane)	1.6	4
Coke-oven gas	3.2	5
Propane-butane mixture	0.6	—
Oil-refinery gas	1.2	—
Shale gas	4.0	—

(5) The Tip-to-Surface Distance. As the ratio of the tip-to-sur-face distance, *h*, to the length, *L*, of the inner cone increases from 0.7 to 2, the effective heat input for tip sizes No. 1 through No. 7
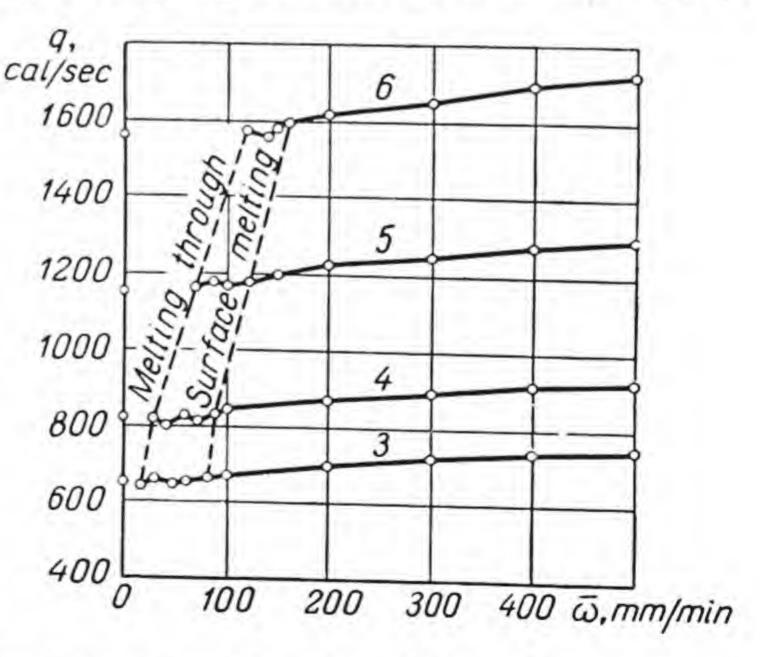
Fig. 71. Effective heat input as a function of the ratio of the tip-to-surface distance, *h*, to the inner cone length, *L*, for tip sizes 1 through 7. For experimental conditions see Fig. 64

falls by as much as 10 to 20 per cent (Fig. 71), and so do the relative and thermal efficiencies and the heating rate.

(6) The Rate of Flame Travel. When the rate of flame travel, $\bar{\omega}$, is increased from zero to 500 mm/min in heating steel plate 6 mm thick, the effective heat input rises by as much as 10 to 15 per cent (Fig. 72). On lighter gauges, this rise is still more

Fig. 72. Effective heat input with tip sizes 3 through 6 plotted against the rate of torch travel. Experimental conditions: $\beta_0 = 1.15$; the other variables same as in Fig. 64

pronounced. It is due to the fact that as the rate of travel increases, the flame comes in contact with an ever colder surface of the metal, and the quantity of heat per unit length of the metal is reduced.

(7) The Thermal Properties of the Metal. Though the thermal diffusivity, a, * increases from 0.09 cm²/sec for steel to 1.1 cm²/sec for aluminium, or twelvefold, the effective heat input increases only by 12 per cent (Fig. 73). Therefore, the thermal efficiency and the welding (heating) rate sharply fall.
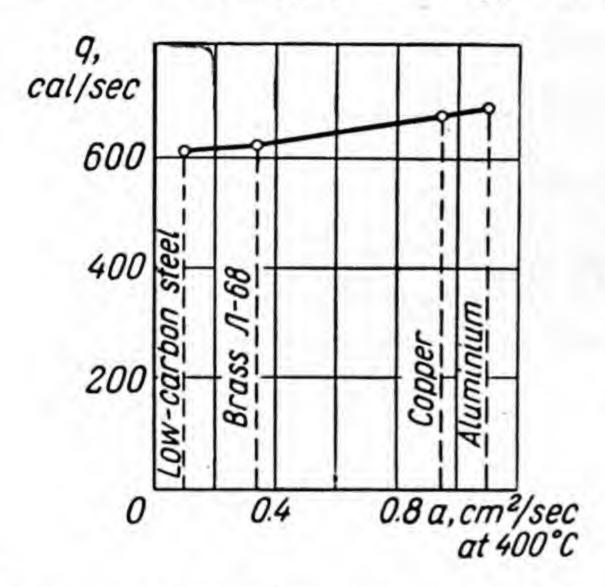
Fig. 73. Effective heat input for tip size No. 3 (400 lit/hr of acetylene) as applied to steel, copper, brass and aluminium, related to their thermal diffusivity at 400° C. Experimental conditions: $\beta_0 = 1.15$; $\bar{\omega} = 350$ mm/min; specimens $110 \times 50 \times 3$ mm; the other variables same as in Fig. 64

The increase in the effective heat input with the higher thermal diffusivity is due to the fall in the temperature of the metal surface brought about by the abstraction of a greater quantity of heat into the body of the metal.

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* Thermal diffusivity, a, is defined as the ratio of thermal conductivity, λ, to the specific heat per unit volume, $c\gamma$, or

$$a = \frac{\lambda}{c\gamma} = \frac{\text{cal/(cm) (sec) (deg. C)}}{\text{cal/(cu cm) (deg. C)}} \text{ or } \text{cm}^2/\text{sec.}$$

Gas welding

CHAPTER IX

WELDING TORCHES

To be good, a torch should be safe in operation and produce a steady flame of constant composition. Manual torches should, in addition, be light in weight and permit of easy control of the flame composition and output in the process of welding.

All existing torches may be classed in a number of ways:

(a) according to the way the mixture is fed into the mixer — into the injector (or low-pressure) type and the pressure type;

(b) according to size and weight — into normal and light-weight or small;

(c) according to the number of flames — into single-flame and multiflame;

(d) according to the fuel gas used — acetylene, hydrogen, petrol, etc;

(e) according to application — manual and machine.

(1) Injector and Pressure Torches for Manual Welding. The most commonly used commercial types in the Soviet Union are ГС and ГСМ * welding torches. Fig. 74 diagrammatically shows an injector torch.

Oxygen under a pressure of 1 to 4 atm (gauge) is taken by the tube 4 to the nozzle 8 in the injector. Issuing from the nozzle 8 at high velocity, the oxygen creates the sucking action which draws the acetylene from the passages in the handle 3 into the mixer 10 to produce the oxy-acetylene mixture. Issuing from the tip 13, the mixture burns to produce the welding flame.

A pressure torch has neither an injector nor a mixer. Instead, it uses a mixing nozzle screwed into the torch head. The torch utilises both gases (oxygen and acetylene) under a roughly equal pressure of 0.5 to 0.7 atm (gauge). The flow rates of the gases

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* These type designations stand for "welding torch" and "small welding torch". respectively.

and the composition of the mixture can be adjusted by the valves and cocks on the torch.

A ГС torch is intended for the manual welding of ferrous and nonferrous metals and can use tip sizes 1 through 7, depending
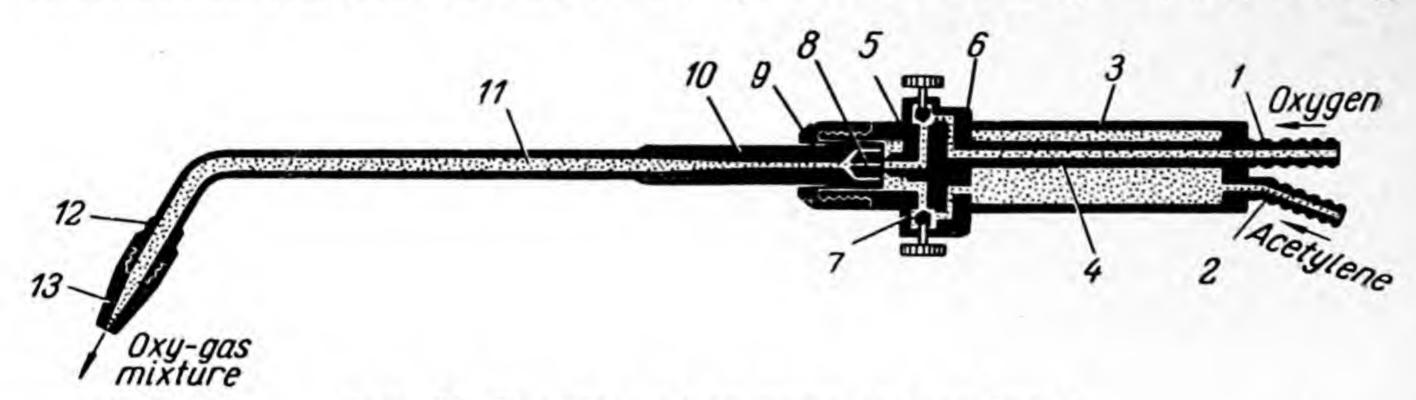
Fig. 74. Diagram of an injector-type torch:
1 — oxygen connection; *2* — acetylene connection; *3* — handle; *4* — oxygen tube; *5* — shank; *6* — oxygen valve; *7* — acetylene valve; *8* — nozzle; *9* — coupling nut; *10* — mixer chamber; *11* — torch head; *12* — coupling nut; *13* — tip

on the metal thickness on hand, which may be from 0.5 to 30 mm. ГС torches available from stock are usually furnished complete

Table 13

Manual Welding Torches

Characteristic	Tip size						
	1	2	3	4	5	6	7
Approximate metal (steel) thickness, mm . . .	0.5-1.5	1-3	2.5-4	4-7	7-11	10-18	17-30
Acetylene flow rate, lit/hr	50-135	135-250	250-400	400-700	700-1,100	1,050-1,750	1,700-2,800
Oxygen flow rate, lit/hr	50-140	140-260	260-420	420-750	750-1,170	1,170-1,900	1,900-3,100
Diameters, mm:							
(a) injector orifice . .	0.25	0.35	0.45	0.6	0.75	0.95	1.2
(b) mixer orifice . . .	0.85	1.25	1.60	2.0	2.5	3.0	3.8
(c) tip	0.80	0.15	1.5	1.9	2.3	2.8	3.5
Oxygen pressure, atm (gauge)	1-4	1.5-4	2-4	2-4	2-4	2-4	2-4
Mixture velocity, m/sec	60-145	70-145	75-145	80-145	90-160	100-165	105-175

with four tip sizes (Nos 3 through 6) for metal thicknesses from 2.5 to 18 mm. The other tip sizes are optional and are furnished on special order. All tip sizes utilise oxygen under a pressure of 1 to 4 atm (gauge) and acetylene over 0.01 atm (gauge).

ГСМ torches go to weld thin-walled articles from ferrous and nonferrous metals 0.5 to 7 mm thick and use tip sizes 1 through 4 which maintain the rate of acetylene flow at 50 to 700 litres per hour, respectively. The performance and other data on ГС and ГСМ torches are summarised in Table 13.

(2) Multiflame Straight-line Torches for the Automatic Welding of Longitudinal Seams. These torches are employed in machines for welding pipes, drum shells and other similar articles.

Fig. 75. Multiflame in-line torch for mechanised welding of longitudinal seams:

1 — oxygen nipple; *2* — acetylene nipple; *3* — shank; *4* — metering washer; *5* — mixer chamber; *6* — casing; *7* — water inlet tube; *8* — water outlet nipple; *9* — union; *10* and *11* — cooling water passage; *12* — water inlet to head; *13* — plugs for nozzle cleaning
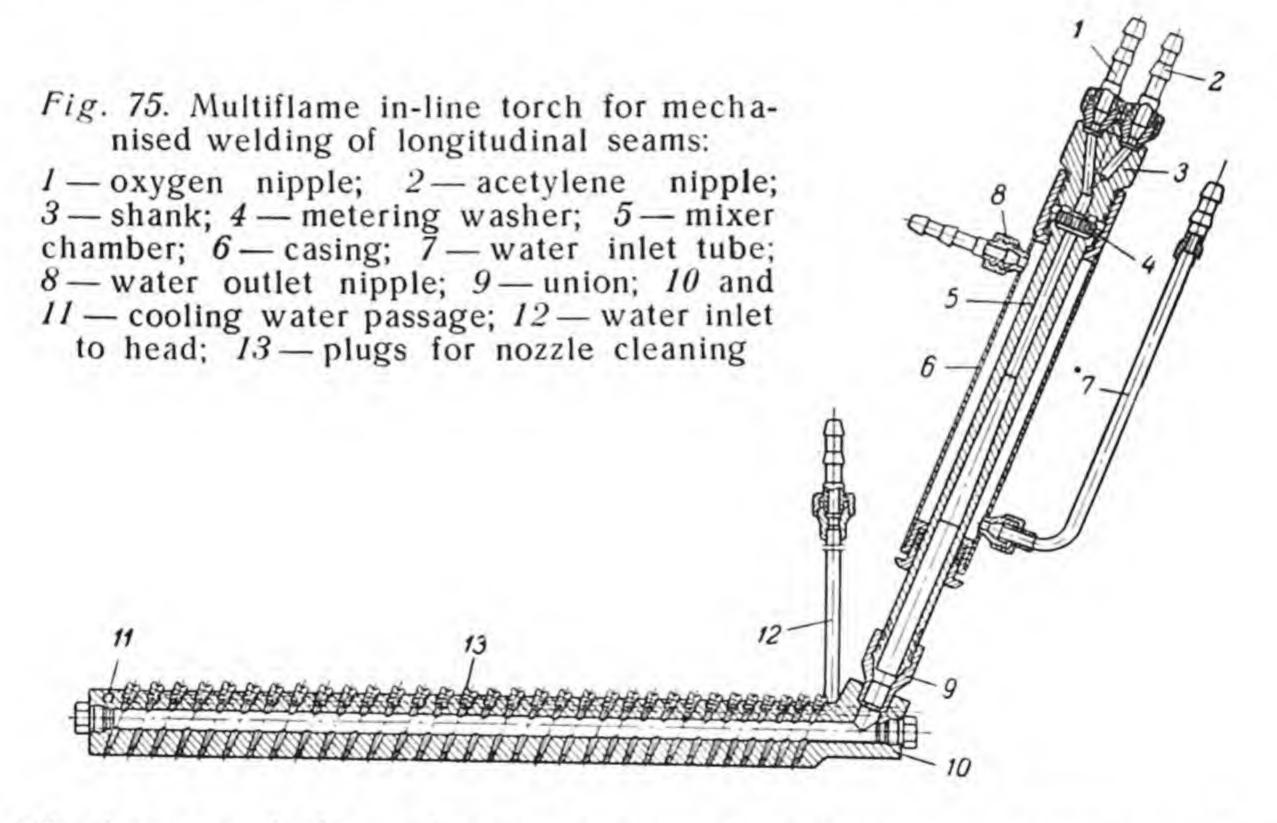
Fig. 75 shows a balanced-pressure torch utilising acetylene and oxygen under a pressure of 0.3 to 0.5 kg/cm². The torch head has 32 jets 1.2 mm in diameter spaced 13 mm apart. The jet axes make an angle of 20° with the vertical. Both the torch head and the mixer are water-cooled. The rate of acetylene flow through the torch is 9 m³/hr.

(3) Automatic Equal-pressure Torches. Where it is essential that the proportion of the gases in the mixture be maintained constant (which may be the case in the welding of nonferrous metals, alloy steels, and in automatic welding), equal (or balanced)

pressure torches should preferably be employed in which the oxygen-to-acetylene ratio in the mixture is set in advance and is maintained constant automatically (Fig. 76).

It is relevant here to discuss the conditions under which the mixture composition can be maintained constant in equal-pressure torches.
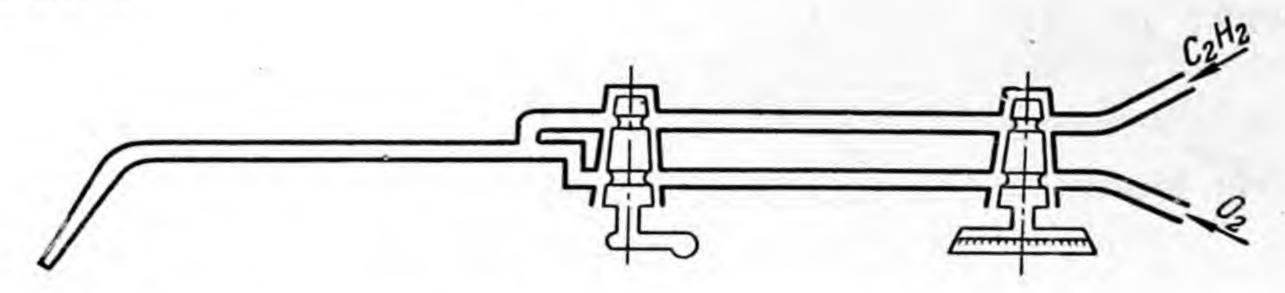
Fig. 76. Diagram of an equal or balanced-pressure torch

The flow of the mixture through the torch may be described by the following equation:

$$V = \mu F w, \qquad (IX.1)$$

where $V =$ gas volume flowing through the tip opening at given pressure and temperature, m³/sec;
$F =$ throat area of the tip, sq m;
$w =$ average gas velocity through the orifices, m/sec;
$\mu =$ coefficient of flow rate.

In equal-pressure torches, the gases issue from the orifices in the torch at critical velocity which, for perfect gases, is described as follows:

$$w_{cr} = \sqrt{2g\frac{k}{k+1}}\sqrt{RT} = \sqrt{2g\frac{k}{k+1}}\sqrt{pv} =$$

$$= \sqrt{2g\frac{k}{k+1}}\sqrt{\frac{p}{\gamma}}, \qquad (IX.2)$$

where $g = 9.81$ m/sec² = acceleration of gravity;
$p =$ gas pressure before the tip throat, kg/m²;

$$k = \frac{c_p}{c_v}$$

$v =$ specific volume of the gas, m³/kg, at pressure p ;
$\gamma =$ specific weight of the gas, kg/m³, at pressure p .

Taking $k = 1.4$ for oxygen and $k = 1.24$ for acetylene, we obtain the critical velocity of their flow.

For oxygen:

$$(w_{ox})_{cr} = \sqrt{2 \times 9.81 \frac{1.4}{1.4+1}} \sqrt{\frac{p_{ox}}{\gamma_{ox}}} =$$

$$= 3.38 \sqrt{\frac{p_{ox}}{\gamma_{ox}}} = 3.38 \sqrt{R_{ox}T_{ox}}.$$

For acetylene:

$$(w_a)_{cr} = \sqrt{2 \times 9.81 \frac{1.24}{1.24+1}} \sqrt{\frac{p_a}{\gamma_a}} =$$

$$= 3.29 \sqrt{\frac{p_a}{\gamma_a}} = 3.29 \sqrt{R_a T_a}.$$

(IX.3)

Substituting w_{cr} from Eq. (IX.3) in Eq. (IX.1), we obtain the rate of gas flow through an equal-pressure torch:

$$\text{For oxygen } V_{ox} = \mu_{ox} F_{ox} \times 3.38 \sqrt{\frac{p_{ox}}{\gamma_{ox}}}.$$

$$\text{For acetylene } V_a = \mu_a F_a \times 3.29 \sqrt{\frac{p_a}{\gamma_a}}.$$

(IX.4)

By definition, the pressure of both oxygen and acetylene before the throat of the respective nozzles must be equal, i. e., $p_a = p_{ox}$. Then we may, for simplicity, consider that $\mu_{ox} = \mu_a$, and substitute $\gamma_{ox} = 1.43$ kg/m^3 and $\gamma_a = 1.17$ kg/m^3 in the respective equation to obtain the requisite oxygen-to-acetylene ratio for an equal-pressure torch:

$$\beta_0 = \frac{V_{ox}}{V_a} = \frac{3.38}{3.29} \times \frac{F_{ox}}{F_a} \sqrt{\frac{\gamma_a}{\gamma_{ox}}} = 0.93 \frac{F_{ox}}{F_a}. \qquad (IX.5)$$

The set proportion of oxygen and acetylene in the mixture will remain constant as long as their pressures remain constant. With a given size of the ports to the mixer, the requisite oxygen-to-acetylene ratio can be obtained by appropriately adjusting the ratio of their pressures, which can be conveniently found from Eq. (IX. 3) and Eq. (IX. 5).

Equal-pressure torches operate from cylinders via a suitable pressure regulator which equalises the pressure of the oxygen and acetylene. For its operation this pressure regulator depends on the pressure of one of the two gases. When the pressure of the reference gas rises, the reducing valve opens to raise the pressure of the other gas on the other side of the diaphragm.

CHAPTER X

FILLER METAL AND FLUXES. STRUCTURAL TRANSFORMATIONS. DEFECTS OF GAS WELDS

1. Filler Metal

Gas welding may be carried out with or without filler metal. When it is used, it may affect the mechanical properties of a gas weld much as some other factors do.

Therefore, filler metal should (a) meet all the requirements of the respective standards to analysis; (b) be of the correct diameter for the thickness of the metal to be welded; (c) be free from scale, rust, grease, paint and other contaminants on the surface; (d) be unable to cause boiling or sputter in the weld pool; and (e) produce a deposit free from blowholes and nonmetallic inclusions.

The major elements present in the welding rod or in the parent metal differ widely in their effect on the weld metal. Some of them improve its mechanical properties, others increase the evolution of gases or the formation of pasty and refractory slag in the puddle and of nonmetallic inclusions in the weld metal.

Carbon reduces the weldability of steel when present in excess of 0.3 per cent. With an increased carbon content, so much gas is evolved in the puddle that the molten metal begins to boil; the grains grow in size; the metal is overheated; the Widmanstätten structure is formed; the hardness is increased and cracks develop in the weld metal and in the heat-affected zone.

Carbon reduces the thermal conductivity of steel, and the heat-affected area becomes wider. This, in turn, produces locked-up stresses in the workpiece. An increase in the carbon content of the welding rod raises the ultimate tensile strength of the weld metal but reduces its ductility. In welding rods for grey cast iron in which graphitisation is mainly dependent on the total content of carbon in the cast iron, the carbon content may be as high as 3.5 or 4 per cent.

Manganese has but an insignificant effect on the weldability of steel when present in low-carbon welding rods in quantities up to 1 per cent. Manganese is an efficient deoxidiser. As much as 1.1 per cent manganese present in the welding rod raises the

ultimate strength of the weld metal to a maximum. When, on the other hand, the increased manganese content in the welding rod is accompanied by an increased carbon content, the weld metal tends to harden and develop cooling cracks. When present in relatively large quantities, the manganese and sulphur in the weld metal form manganese sulphide (MnS) which has a high melting point and dissolves little in the molten iron. For this reason, it may remain in the weld metal as scattered slag inclusions. Yet, some manganese is sometimes added to the weld metal in order to control its susceptibility to hot cracking provoked by the sulphur. When it is combined in high-melting manganese sulphide, there is little chance for the sulphur to segregate. The proportion of manganese in welding rods for low-carbon steel should be kept at, or in excess of, 30 per cent of the sulphur present. In welding rods for grey cast iron the manganese content is usually limited to 0.5 or 0.6 per cent, as it has a chilling effect on cast iron.

Silicon is an undesirable impurity in low-carbon welding rods for steel, since, when it is present in increased quantities, silicates like $FeO \cdot SiO_2$ and $MnO \cdot SiO_2$ are formed, which appear as a high-melting thick and pasty blanket of slag over the puddle. This slag prevents the gases from escaping when the puddle solidifies. As a result, the weld metal will contain a large number of gas pockets and nonmetallic inclusions. It is for this reason that, although silicon is a good deoxidiser, its content in low-carbon welding rods in limited to 0.03 per cent. In welding rods for grey cast iron, on the contrary, the content of silicon should be as high as practicable (3 to 3.5 per cent), as silicon promotes graphitisation, which is essential if good grey cast iron is to be obtained in the weld metal.

Sulphur in steel readily combines with the iron to form iron sulphide (FeS) which has a tendency to segregate and form brittle networks at the grain boundaries. On account of its low melting point, the iron sulphide causes a lack of cohesion between adjacent grains of the metal when it is heated above a red heat (red shortness). The sulphur content in filler metal should not exceed 0.04 per cent. In cast iron sulphur is likewise an undesirable impurity. Its content in welding rods for grey cast iron is limited to 0.08 per cent. Sulphur hinders the precipitation of graphite in cooling, raises the melting point of cast iron and aggravates porosity in the weld metal. In copper, sulphur forms Cu_2S which produces gaseous SO_2 in welding, also responsible for porosity in the copper weld.

Phosphorus in steel produces cold shortness, decreases its plasticity, and brings about lack of homogeneity in the metal. The

content of phosphorus in welding rods for steel should not exceed 0.04 per cent. In cast iron, phosphorus (when 0.5 to 1 per cent of it is present) is a useful element, as it improves the fluidity of molten cast iron, and the molten metal fills the groove better. The phosphorus content in welding rods for cast iron should be anywhere between 0.5 and 0.8 per cent. Phosphorus is also a useful element in welding copper and its alloys (bronzes and brasses), as it is a good deoxidiser when present in copper wire in 0.25 to 0.4 per cent.

Nickel increases the strength, hardenability and, to an extent, ductility of steel. In most grades of welding rods for plain carbon steels nickel is present by chance, and its content does not exceed 0.25 or 0.3 per cent. In welding rods for stainless and heat-resistant Cr-Ni steels nickel may run as much as 8 to 20 per cent. In these steels, nickel is added to obtain a stable austenitic structure.

Chromium sharply increases the hardenability of steel. In most grades of welding rods for plain carbon steels, chromium, like nickel, is not introduced on purpose. Its content is limited to 0.1 or 0.2 per cent. In welding rods for low-alloy chrome-molybdenum and cromansil steels the chromium content should be 0.8 to 1.1 per cent. For high-alloy chrome-molybdenum steels (such as X5M) its content may be as high as 5 to 7 per cent. The chromium content is especially high (12 to 27 per cent) in welding rods for high-chromium heat- and corrosion-resistant steels.

In view of the fact that chromium reduces the ductility of steel, it is introduced in company with other alloying elements, such as nickel or molybdenum.

Molybdenum affects the mechanical properties of steel even when present in small quantities (as low as 0.15 to 0.3 per cent). It appreciably raises the hardness and strength of steels, especially at high temperatures, eliminates the brittleness of chrome steels at intermediate temperatures and improves the toughness of steels at both low and high temperatures. Welding rods for chrome-molybdenum steels may contain as much as 0.15 to 0.6 per cent molybdenum, and more (2 or 3 per cent) for special-purpose chromium-nickel-tungsten steel.

Under the relevant U.S.S.R. State Standard, welding wire for steels is available in diameters from 0.3 to 12 mm.

Wire 0.3 to 2 mm in diameter is supplied in coils, each coil weighing at least 20 kg when made from plain carbon steel; 2 to 15 kg when made from alloy steel, and 1.5 to 8 kg when made from high-alloy steel. For wire 2.5 to 12 mm in diameter, the weight of a coil is at least 40 kg in the case of low-carbon steel,

at least 20 kg for alloy steel wire, and at least 10 kg for high-alloy steel wire.

Until recently, only low-carbon welding wire has been used to gas-weld low-carbon steels. This wire cannot produce full-strength welds and is therefore unsuitable for important applications. A more recent trend has, therefore, been towards using low-alloy filler metal for this purpose, such as silicon-manganese and manganese welding rods. Their use has been promoted by the Autogenous Welding Research Institute. Apart from raising the strength of welded joints, these rods possess better workability properties.

Low-alloy steels may be successfully welded with low-alloy chromium welding rods, especially where it is important that the weld metal have an ultimate tensile strength of at least 45 to 48 kg/mm².

Tables 14 and 15 give the chemical compositions of the welding rods for low-carbon and low-alloy steels, investigated by the Autogenous Welding Research Institute, and the mechanical properties of the welded joints obtained.

Table 14

Welding rods	Chemical composition						
	C	Si	Mn	Cr	Ni	P	S
Low-carbon steel . . .	0.08	trace	0.36	0.13	0.013	0.24	0.008
Manganese steel	0.14	0.12	0.81	trace	0.25	0.023	0.032
Manganese steel	0.17	0.29	1.23	0.14	0.20	0.20	0.025
Chromium steel	0.24	0.21	0.42	0.96	0.17	0.03	0.35
Silicon-manganese steel	0.10	0.65	0.97	trace	0.10	0.0040	0.024

Table 15

Welding rods	U. T. S., kg/mm²		Per cent elongation		Impact strength, kg-m/cm²	
	as welded	normalised by torch	as welded	normalised by torch	as welded	normalised by torch
Low-carbon steel	33-35	35-37	9-14	15-19	1.5-3	5-13
Manganese steel	37-40	36-43	12-15	13-18	3.5-12	10-15
Manganese steel	44-48	30-34	22-27	6-6.5	—	—
Chromium steel	49-52	43-44	3-10	12-18	2-6.5	12-17
Silicon-manganese steel	45-48	45-48	10-13	13-17	5-8	11-15

Table 16

Grade	Chemical composition, per cent													Suggested application
	Cu	P	Si	Sn	Bi	Sb	As	Fe	Ni	Pb	S	O	Zn	
M1	99.90	—	—	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.005	Max. 0.002	Max. 0.005	Max. 0.005	Max. 0.08	Max. 0.005	Gas welding of M1, M2 and M3 copper
Мф-03	99.7-99.5	0.2-0.4	—	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.005	Max. 0.002	Max. 0.005	Max. 0.005	Max. 0.08	Max. 0.005	Gas welding of M1, M2 and M3 copper up to 10 mm thick
МФК-03-02	99.2-99.6	0.2-0.4	0.15-0.3	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.005	Max. 0.002	Max. 0.005	Max. 0.005	Max. 0.08	Max. 0.005	Same, for over 10 mm and all-position welding
Л62	60.5-63.5	—0.01	—	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.002	Max. 0.005	Max. 0.002	Max. 0.005	Max. 0.005	Max. 0.08	Max. 0.005	Gas welding of brass Л62 with БМ-1 vapour flux
ЛК62-05	60.5-63.5	—0.01	0.3-0.7	—	Max. 0.002	Max. 0.15	—	Max. 0.15	—	Max. 0.08	—	—	Balance	Gas welding of brass Л62
ЛО60-1	59.0-61.0	—0.01	—	1-1.5	Max. 0.002	Max. 0.005	—	Max. 0.10	—	Max. 0.3	—	—	Balance	Gas welding of brass ЛО62-1 with БМ-1 vapour flux
ЛОК59-1-03	58.0-60.0	—0.01	0.2-0.4	0.7-1.1	Max. 0.003	Max. 0.01	Max. 0.01	Max. 0.15	—	Max. 0.1	—	—	Balance	Gas welding of brass ЛО62-1

As will be seen from Table 15, normalising, which can be conveniently applied by using the torch, markedly improves the mechanical properties of the deposited metal and may therefore be recommended for low-carbon and low-alloy steels.

The best results in gas-welding low-carbon steels have been obtained with silicon-manganese welding rods which produce welded joints comparable in strength with those usually made by the arc process using shielded-arc electrodes.

For steels with the medium and medium-to-high content of the alloying elements, the welding wire should be of the same analysis as the parent metal.

According to the relevant standard, the welding rods for grey cast iron in the Soviet Union are available in two grades, namely:

1. grade A rods for welding heavy iron castings with a preheat, containing 3-3.6 per cent C, 3-3.5 per cent Si, 0.5-0.8 per cent Mn, 0.2-0.5 per cent P, and not over 0.08 per cent S;

2. grade B rods for welding smaller castings with a local preheat and a high rate of post-welding cooling, containing 3.6-4.8 per cent Si, 0.3-0.5 P, and as much the other elements as in Grade A rods.

The two grades of welding rods come in diameter sizes of 6, 8, 10, and 12 mm.

Table 16 presents some of the welding rod grades for copper and brass welding.

Bronzes, aluminium and its alloys, nickel, lead and some other metals should be welded with rods or wire of the same composition as the parent metal.

In the light of field experience, the Autogenous Welding Research Institute has drawn up specifications covering filler metal and fluxes for oxy-acetylene surfacing — a process by which a good deal of work is being done. Table 17 lists the principal characteristics of such rods.

2. Fluxes

The metallurgical side of gas welding differs from the ordinary metal-making process in that:

(a) the bath of molten metal is small in volume;

(b) the metal is heated locally by a travelling source of concentrated and intense heat;

(c) the source of heat has a high temperature;

(d) the rates of both heating and cooling are high;

(e) the molten metal reacts with the hot gases in the intermediate (reducing) zone of the flame;

Table 17

Surfacing material	Grade designation	Rod dia, mm	Rod length, mm	Properties of deposit				Field of application
				Max. hardness	Machinability	Red hardness, °C	Corrosion resistance	
Air-hardening alloy steel	3Х2В8	3 5 8	600±6 600±6 1,000±10	22	Good	Up to 500	None	Surfacing of dies and punches employed at elevated and sharply varying temperatures
Ditto	Х9С2	1 3 5	600±6 600±6 1,000±10	22	Good	Up to 500	Fair	Surfacing of aircraft engine valves and other parts subjected to elevated temperatures at low loads
Ditto	Св18ХГСА	3 5 7	600±6 600±6 1,000±10	22	Good	None	None	Reclamation of parts from respective steel grades (aircraft and automotive applications)
Ditto	Р9	1 3 5 7	600±6 600±6 600±6 1,000±10	28	Fair	Up to 600	None	Fabrication and reconditioning of high-speed cutting tools
Ditto	Р18	1 3 5 7	600±6 600±6 600±6 1,000±10	28	Fair	Up to 600	None	Same, with better properties

Alloy group	Alloy name	Layer	Min. hardness	Rockwell	Machinability	Max. temp.	Corrosion resistance	Application
Cast carbide alloys	No. 2 Sormite	3 4 5 6 7 8	Min. 200 Min. 300 Ditto Ditto Min. 500 Ditto	35	Fair	None	None	Surfacing of parts subjected to impact at normal temperatures
Ditto	No. 1 Sormite	Ditto	Ditto	45	Grinding only	Up to 600	Limited	Surfacing of cutting tools for hot work and cold work in absence of shock loads
Ditto	BK	4 5 6 8	Min. 350 Ditto Min. 500 Ditto	40	Ditto	—	None	Surfacing of parts subjected to abrasion, such as plough shears, dipper teeth, etc.
Ditto	X4	Ditto	Ditto	46	Ditto	—	Limited	Same, when abrasion is severe
Iron-base hard-facing alloys	Stellite B2K	3 4 5 6 7 8	Min. 200 Ditto Min. 300 Ditto Min. 300 Ditto	46	Ditto	Up to 750	Good	Reclamation of parts subjected to wear due to corrosion and elevated temperatures, such as turbine blades, engine valves, etc.
	Stellite B3K	Ditto	Min. 300	41	Ditto	Ditto	Ditto	Same. Higher resistance to impact, but lower wear resistance

(f) the molten metal is vigorously puddled by the hot gases of the flame and the welding rod.

The fact that the welding operation lasts very little time and the bath is small in volume makes it difficult to follow the reactions closely. The judgement of the metallurgical processes taking place in the molten pool is an approximate one, based mainly on the analysis of the initial and final state of the system and, in part, on the analogy with ordinary metal-making.

In welding, the molten metal of the weld comes in contact with the gaseous phase of the intermediate flame zone carrying mainly CO and H_2, although water vapour, CO_2, H, O_2 and N_2 are also present there. The intermediate zone may also carry a negligible quantity of free carbon which has failed to be oxidised to CO at the boundary of the inner cone.

The nature of the reactions in the molten pool is mainly determined by the composition of the intermediate flame zone which is in turn dependent on the proportion of oxygen and fuel gas in the mixture. The principle reactions in the molten metal are those of oxidation and reduction.

From the theory of metallurgy it is known that the oxides of metals with a strong affinity for oxygen have a lower dissociation pressure, are more stable and more difficult to reduce to metallic form.

The reducing agent for a given metal should have a greater affinity for oxygen than the metal, and the oxide of the deoxidiser should have a lower dissociation pressure than the oxide to be reduced.

Table 18 presents some of the oxides of the most important elements in the order of increasing dissociation pressure.

Referring to Table 18, the highest dissociation pressure is shown by copper oxide (Cu_2O) and the lowest by calcium oxide

Table 18

Oxide	Element	Oxide	Element
CaO	Ca	P_2O_5	P
MgO	Mg	FeO, WO_2	Fe, W
Al_2O_3, ZrO_2	Al, Zr	MoO_2	Mo
TiO_2	Ti	CoO	Co
SiO_2	Si	NiO	Ni
MnO, Cr_2O_3	Mn, Cr	As_2O_3	As
ZnO	Zn	Cu_2O	Cu

(CaO). Fig. 77 relates the dissociation pressure of oxides to temperature.

In the case of copper, nickel and iron, no oxides are practically formed in the weld as the CO and H_2 of the intermediate zone easily reduce them. This is not the case, however, with magnesium,
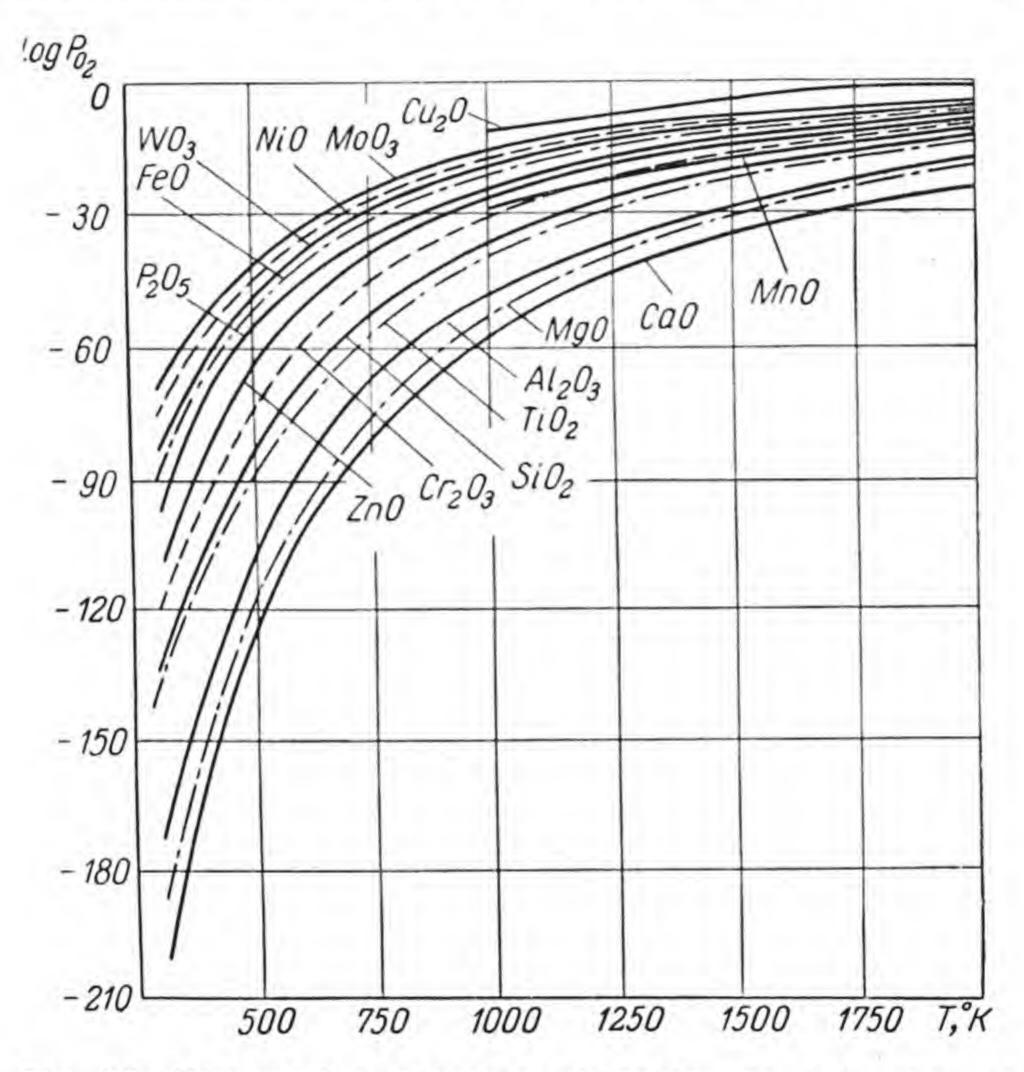
Fig. 77. Dissociation pressure of oxides as a function of temperature

aluminium and zinc, which fact necessitates the use of some cleansing agents capable of either combining with the oxides or dissolving them.

These cleansing agents are called fluxes. Appart from dissolving oxides, they serve to release trapped gases and slag and to remove nonmetallic inclusions. Moreover, fluxes form a blanket of molten slag over the molten pool, thereby preventing the metal from further oxidation.

The use of fluxes in the case of some high-alloy steels, cast iron and nonferrous metals and alloys is necessitated by the fact that a film of oxide is always formed (especially, at elevated tem-

peratures) on their surface, which, if carried over into the weld metal, will prevent a reliable union between the parent and filler metal. In welding low-carbon steels this film of oxide can be easily removed by puddling and reduced by the flame.

The principal requirements a good flux should meet are:

(1) a flux should easily melt and have a melting point lower than the parent and filler metal;

(2) a flux should readily react with metallic oxides so that the oxides are completely dissolved by the time the molten pool solidifies. The blanket of slag should reliably protect the metal from atmospheric oxidation and come off easily upon solidification;

(3) a flux should have no injurious effect on the metal; its specific gravity should be lower than that of the metal so that the slag would easily float to the surface of the bath;

(4) the molten flux should readily spread over the molten metal.

The oxides and nonmetallic inclusions can be removed from the puddle either chemically or by dissolving them. In both cases the effect of a flux obeys the distribution law under which a solute tends to distribute in equilibrium concentration between the two phases of a mixture of immiscible solvents at a constant temperature.

Either a basic or an acid flux should be used, depending on the nature of the oxides formed. In both cases the reaction will be:

$$acid\ oxide + basic\ oxide = salt.$$

Where the oxides present in the puddle are mainly basic (which is true of most metallic oxides *), an acid flux should be used, and vice versa.

Acid fluxes are usually employed in welding nonferrous metals, especially copper-base ones (such as copper, brasses and bronzes). Basic fluxes are chiefly used in cast iron welding.

The fluxes used to weld copper-base alloys are for the most part boron compounds, such as borax ($Na_2B_4O_7 \cdot 10H_2O$), boric acid (H_3BO_3), or their mixtures. Before use, borax should be calcined; if not, it will produce blisters and give off crystallisation water. If this were allowed to take place in the puddle, the quantity of water vapour in the welding zone would grow undesirably high. In addition, the borax would come off the metal in giving up the water of crystallisation, and the metal would be exposed to atmospheric oxidation.

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* Except for TiO_2 which is an acid oxide.

The reaction between borax and copper oxide may be thought of as follows.

When heated, the borax dissociates into sodium metaborate ($NaBO_2$) and boric anhydride (B_2O_3):

$$Na_2B_4O_7 \rightarrow 2NaBO_2 + B_2O_3, \qquad (X.1)$$

which combine with the copper to form sodium-copper metaborate:

$$2NaBO_2 + B_2O_3 + CuO = (NaBO_2)_2 \cdot Cu(BO_2)_2. \qquad (X.2)$$

The reaction between borax and zink oxide proceeds in a similar way:

$$2NaBO_2 + B_2O_3 + ZnO = (NaBO_2)_2 \cdot Zn(BO_2)_2. \qquad (X.3)$$

When the flux is boric acid, it should be remembered that, when heated to 550° or 600° C, it gives up all water and is turned to boric anhydride (B_2O_3) which is acidic in nature. It combines with the welding oxides to form a variety of borates, such as $CuOB_2O_3$ in copper welding.

For the latter case the concentration of CuO in the metal can be easily found by the distribution law. The distribution coefficient will be

$$L = \frac{(CuO)_{slag}}{[CuO]_{metal}}. \qquad (X.4)$$

If to the slag of this system "slag-metal oxide" an oxide is added, reacting with the slag according to the equation

$$CuO + B_2O_3 \rightleftarrows CuO \cdot B_2O_3 \qquad (X.5)$$

and having an equilibrium constant

$$K_{eq} = \frac{(CuO)(B_2O_3)}{(CuO \cdot B_2O_3)}, \qquad (X.6)$$

the concentration of free oxide in the slag will be

$$(CuO) = \frac{(CuO \cdot B_2O_3)}{(B_2O_3)} K_{eq}, \qquad (X.7)$$

whence

$$L = \frac{(CuO \cdot B_2O_3) K_{eq}}{(B_2O_3)[CuO]} \qquad (X.8)$$

or

$$[CuO] = \frac{(CuO \cdot B_2O_3) K_{eq}}{L(B_2O_3)}. \qquad (X.9)$$

From Eq. (X.9) it follows that the concentration of CuO in the metal decreases with increasing B_2O_3 content and with decreasing equilibrium constant, K_{eq}. The value of K_{eq} decreases as

12—413

the dissociation level of the compound in the slag grows lower.

When using borax and boric acid, it should be remembered that in solidifying borates form a glass-like crust on the weld, which comes off with difficulty. To prevent this, it is good practice to add some potassium carbonate (K_2CO_3) or sodium phosphate (Na_2HPO_4) to the borax and boric acid to improve the fluidity of the slag and to make the oxides more soluble in borax or boric acid (Table 19).

Table 19

Most Commonly Used Fluxes for Welding Copper and Copper-base Alloys

(in per cent by weight)

Boric acid, H_3BO_3	Borax, $N_2B_4O_7$	Sodium phosphate, Na_2HPO_4	Potassium carbonate, K_2CO_3	Sodium chloride, NaCl
100	—	—	—	—
—	100	—	—	—
50	50	—	—	—
25	75	—	—	—
35	50	15	—	—
—	56	—	22	22

When borax and other powdered fluxes are added to the puddle by hand in unequal amounts, it is impossible to ensure steady deoxidation of the metal in welding, and the quality of welded joints is therefore not sufficiently high.

A more recent trend has been to feed vapour and powdered fluxes to the welding zone together with the fuel gas in precisely metered quantities.

The Autogenous Welding Research Institute has developed a vapour flux for the welding of brasses and copper-base alloys. It is an azeotropic mixture (i. e., a mixture having a constant boiling point) consisting of 75.5 per cent tetramethylborate $(CH_3O)_3B$ and 24.5 per cent methanole (CH_3OH) fed to the torch by a suitable mechanism.

The liquid azeotropic mixture is produced in a rectification still in which sulphuric acid is allowed to react with commercial borax and methanole:

$$Na_2B_4O_7 \cdot 10H_2O + 16CH_3OH + 2H_2SO_4 \rightarrow 2NaHSO_4 +$$
$$+ 4[(CH_3O)_3B + CH_3OH] + 17H_2O \qquad (X.10)$$

The gaseous fraction drawn from the still for the first six hours is a mixture of methanole vapours with methylborate. The condensate of this fraction is an azeotropic mixture with a boiling point of 54° C.

The liquid thus obtained is poured into the flux dispenser where it soaks a wick through which acetylene is fed from a cylinder or a generator to the torch. The acetylene picks up vapours of the azeotropic mixture and adds them to the welding flame in the requisite quantity, giving the flame a greenish tinge.

No boric anhydride (B_2O_3), as such, is fed to the flame, while boron is present there in the form of tetramethylborate $(CH_3O)_3B$. From this tetramethylborate the heat of the flame produces boric anhydride which combines with the copper and zinc oxides much as in welding with powdered borax and boric acid.

In cast iron welding, basic fluxes (containing primarily Na_2CO_3, $NaHCO_3$ and K_2CO_3) should be used, though both acidic and basic fluxes may also be employed, provided that the basic oxides are prevalent.

This requirement is essential in order that the film of silicon oxide (SiO_2) usually formed on cast iron can be dissolved, thereby making the molten slag more fluid, reducing its specific gravity and causing it to float to the surface of the molten bath.

The sodium carbonate contained in the flux reacts with the silicon oxide as follows:

$$2Na_2CO_3 + SiO_2 = Na_4SiO_4 + 2CO_2. \qquad (X.11)$$

Pure borax ($Na_2B_4O_7$) may also be used as flux for cast iron.

The fluxes most commonly used in cast iron welding are presented in Table 20.

Table 20

Most Commonly Used Fluxes for Cast Iron Welding

(per cent by weight)

Borax ($Na_2B_4O_7$)	Sodium carbonate (Na_2CO_3)	Potassium carbonate (K_2CO_3)	Sodium nitrate ($NaNO_3$)	Sodium bicarbonate ($NaHCO_3$)
100	—	—	—	—
56	22	22	—	—
23	27	—	50	—
—	50	—	—	50

In some cases, especially in welding aluminium alloys, the deoxidation has to be supplemented by mechanical dissolution of

the oxides formed in welding. For this purpose, a flux should additionally contain some salts capable of dissolving oxides in a molten state.

For example, aluminium oxide (Al_2O_3) dissolves in molten cryolite ($AlF_3 \cdot 3NaF$), although Al_2O_3 is mainly dissolved chemically.

Fluxes for aluminium welding are usually mixtures of fluorides and chlorides of lithium, potassium, sodium and calcium (see Table 21).

Table 21

Most Commonly Used Fluxes of Aluminium Welding

(per cent by weight)

Ingredient	Flux designation							
	1	2	3 Аф-4А	4	5	6	7	8
Sodium chloride	30	45	28	35	33	19	30	41
Potassium chloride . . .	45	30	50	48	45	29	45	51
Lithium chloride	15	10	14	9	15	—	15	—
Barium chloride	—	—	—	—	—	48	—	—
Sodium fluoride	—	—	8	8	—	—	10	8
Potassium fluoride . . .	7	15	—	—	7	—	—	—
Fluorspar	—	—	—	—	—	4	—	—
Sodium bisulphate . . .	3	—	—	—	—	—	—	—

Aluminium oxides are best dissolved by lithium salts, mainly lithium chloride which is present in almost all of the fluxes used in aluminium welding. Al_2O_3 reacts with lithium chloride as follows:

$$6LiCl + Al_2O_3 \rightleftharpoons 2AlCl_3 + 3Li_2O. \qquad (X.12)$$

Fluxes for aluminium welding, containing LiCl, are hygroscopic and must therefore be stored in hermetically sealed cans.

After welding it is important to remove all surplus flux from and near the weld with a weak (2-per cent) solution of nitric acid and rinse the weld with water. Otherwise, the further reaction between the LiCl and Al_2O_3 will corrode the metal surface.

Fluxes are also employed in some of the applications of flame hard-facing. Their compositions are summarised in Table 22.

Table 22

Fluxes for Flame Hard-facing

Hard-facing material	Flux		Application
	Ingredients	per cent	
3X2B8; 18XΓCA; X9C2; P-9; P-18	Calcined borax . . . . Boric acid	50 50	In powdered form, applied as a thin coating to the surface to be hard-faced. In hard-facing, flux is scooped by the end of hard-facing rod
X9C2	Marble . . . . Fluorspar . . . . Ferro-titanium . . . . . Ferro-silicon . . . . . . Ferro-manganese . . . . Borax . . . . Sodium bisulphate . . .	48.0 24.0 8.8 4.8 4.8 5.6 4.0	In powdered form or as a coating applied to hard-facing rod
P-9	Marble . . . . Fluorspar . . . . Quartz sand . . . . . Ferro-manganese (low-carbon) . . . . Ferro-silicon . . . . . . Ferro-titanium . . . . . Water glass (dry silicate in the charge)	54.0 16.0 9.0 5.0 5.0 12.0 10-15 (by weight of the other ingredients)	As a thin coating applied to hard-facing rod
White cast iron KK Chromium cast iron XЧ	Dehydrated borax . . . Sodium carbonate . . . Sodium nitrate	23.0 27.0 50.0	In powdered form, applied as a thin coating to the surface to be hard-faced. In hard-facing, flux is scooped by the end of hard-facing rod

Table 22 (continued from page 181)

Hard-facing material	Flux		Application
	Ingredients	per cent	
Nos. 1 and 2 Sormite	Borax	50.0	In powdered form
	Sodium bicarbonate . .	47.0	
	Silica	3.0	
B2K and B3K Stellite	*For carbon steel*		
	Dehydrated borax . . .	100.0	In powdered form
	For alloy steels		
	Boric acid	68.5	Same
	Dehydrated borax . . .	20.0	
	Fluorspar	11.5	

3. Structural Transformations in the Weld and the Heat-affected Zone

With much heat imparted to the workpiece in gas welding and the flame covering a considerable width of the parent metal, the weld metal cools at a slower rate and there is no preferable path for heat abstraction from the solidifying puddle as in arc welding. For these reasons, the gas weld has a coarse-grained structure with more uniform grain size and a more pronounced intercrystalline liquation. The columnar structure, typical of arc welds, is formed in gas welding only in metal having considerable thermal conductivity.

In the case of steel containing 0.15 to 0.3 per cent carbon, the weld metal may have a Widmanstätten structure in which the ferrite plates make angles of 60, 90 and 120° with the crystallographic axes of the austenite grains. This, however, is the case only when the metal has a coarse-grained austenitic structure and the rate of cooling is high in the transition range from Ar_3 to Ar_2.

The size and shape of the crystallites in the weld metal are greatly dependent on how much the puddle has been superheated, on the rate of cooling and the puddling action of the flame, welding rod and the gases evolved by the molten metal. The grain is finer and the weld metal has higher mechanical properties when the amount of superheat has been insignificant and the rate of cooling is high. Therefore, the speed of gas welding should be as high as practicable.

The heat-affected zone in gas welding has the same areas as in arc welding, but the various areas in this zone are wider due to the fact that the flame is less concentrated than the arc and the heating time is longer.

The width of the heat-affected zone varies with the thickness of the metal, the welding technique and the welding conditions. In heavy-gauge plate the heat-affected zone is narrower when welding by the rightward than by the leftward technique. On the other hand, on lighter gauges the rightward technique produces a broader heat-affected zone than leftward welding. The welding conditions are mainly determined by the size of the tip and the rate of flame travel.

The heat-affected zone will be of the least width when the tip size has been chosen properly and the welding speed is sufficiently high. Depending on these variables, the width of the heat-affected zone may range from 5 to 30 mm.

The structural transformations occurring in the heat-affected zone are mainly determined by the grade of metal, welding conditions and welding technique. In pure metals and single-phase alloys which undergo no allotropic transformations, the welding heat only gives rise to grain growth. In complex metals and alloys this same heat brings about additional phase and structural changes.

In a general case, the heat-affected zone in a low-carbon or a low-alloy steel includes the following areas.

The area of partial melting is the fusion area between the weld metal and the parent metal. In heating, complete recrystallisation takes place in this area, accompanied by a vigorous grain growth and partial fusion of their boundaries.
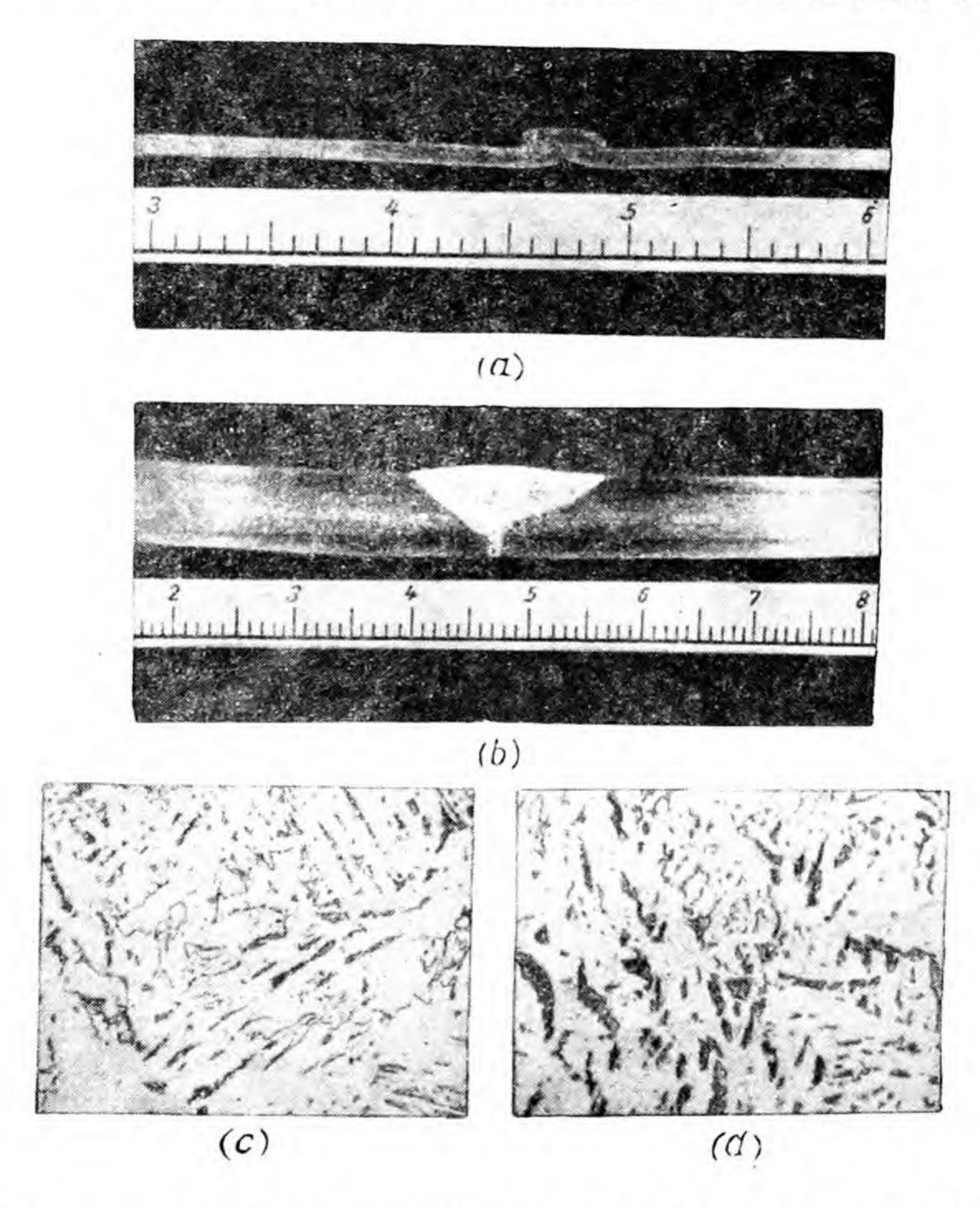
The area of grain growth and superheat includes the metal layers raised to a temperature close to the melting point of the steel. This area is characterised by a coarse-grained structure and, in some cases, the Widmanstätten structure reducing the ductility of the steel. This area is most dangerous in air-hardening steels, as the superheat and subsequent quenching reduce their ductility more than in any other steel. This area, customarily called the near-weld area, is appreciably larger in gas welding than in arc welding.

The area of complete recrystallisation, as the two former areas, is heated to above Ac_3 but below the temperature at which the grains begin to grow intensively. Therefore, the recrystallisation that takes place in this area produces a refined structure. In some cases, however, mainly in heavy-gauge steel plate, when the metal has to be held at this temperature for a long time, a secondary

grain growth may occur with the resultant open-grain structure.
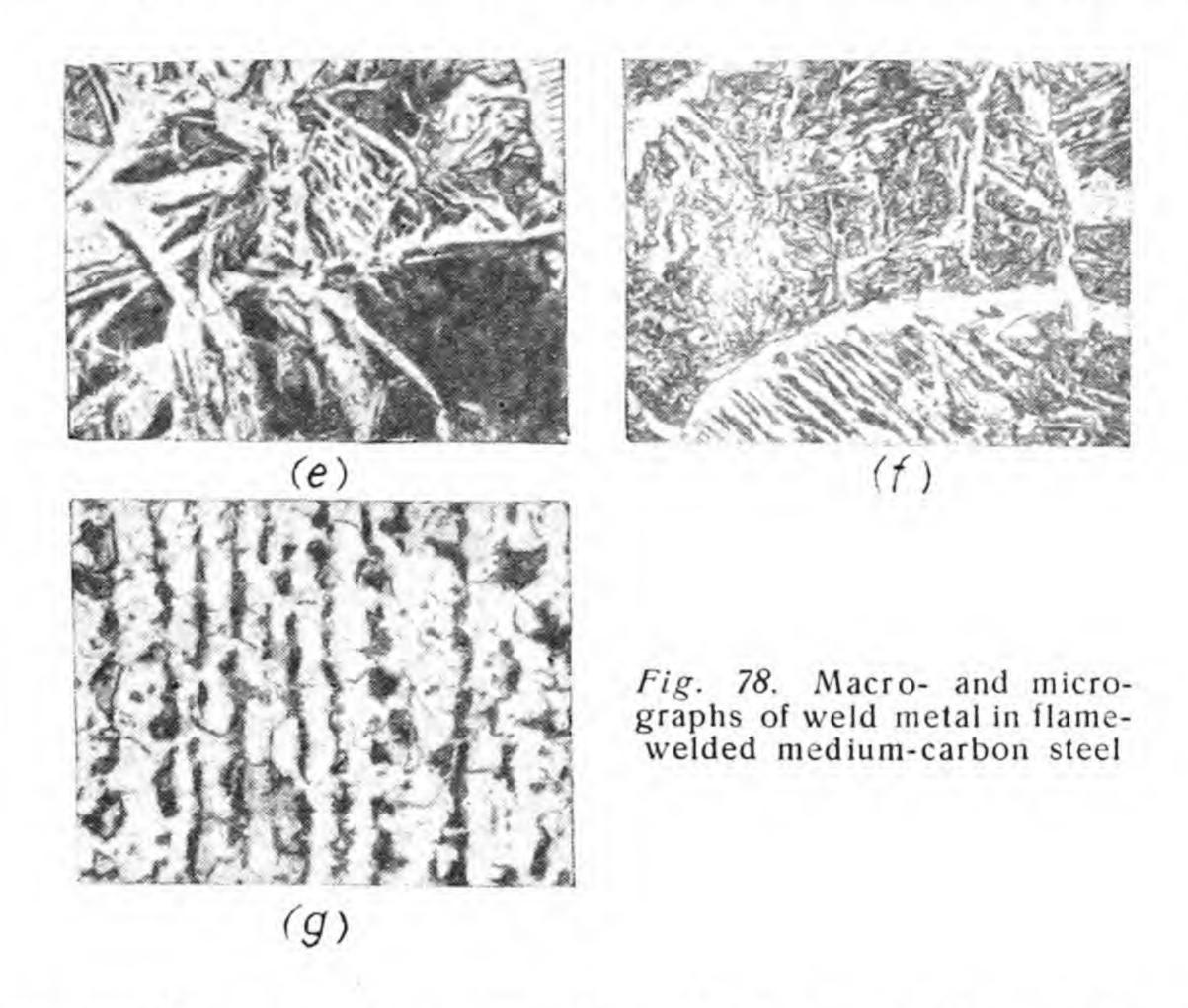
The area of incomplete recrystallisation is an area where the metal is heated to anywhere between Ac_1 and Ac_3 and has a rather

(a)

(b)

(c)

(d)

heterogeneous structure consisting of the primary coarse ferrite grains and the finer grains produced by recrystallisation.

The recrystallisation or temper area encompasses the metal heated from 200-400° C to Ac_1. The recrystallisation, which takes the form of the transition of atoms from lattice to lattice and is accompanied by grain growth, may only occur when the metal

has undergone plastic deformation (forging, cold rolling, machining, shearing and slitting) which destroys the outer shells, crushes the grains, and displaces or rotates the grain fragments, thereby facilitating the transition of iron atoms from grain to grain and

(e)

(f)

(g)

Fig. 78. Macro- and micrographs of weld metal in flame-welded medium-carbon steel
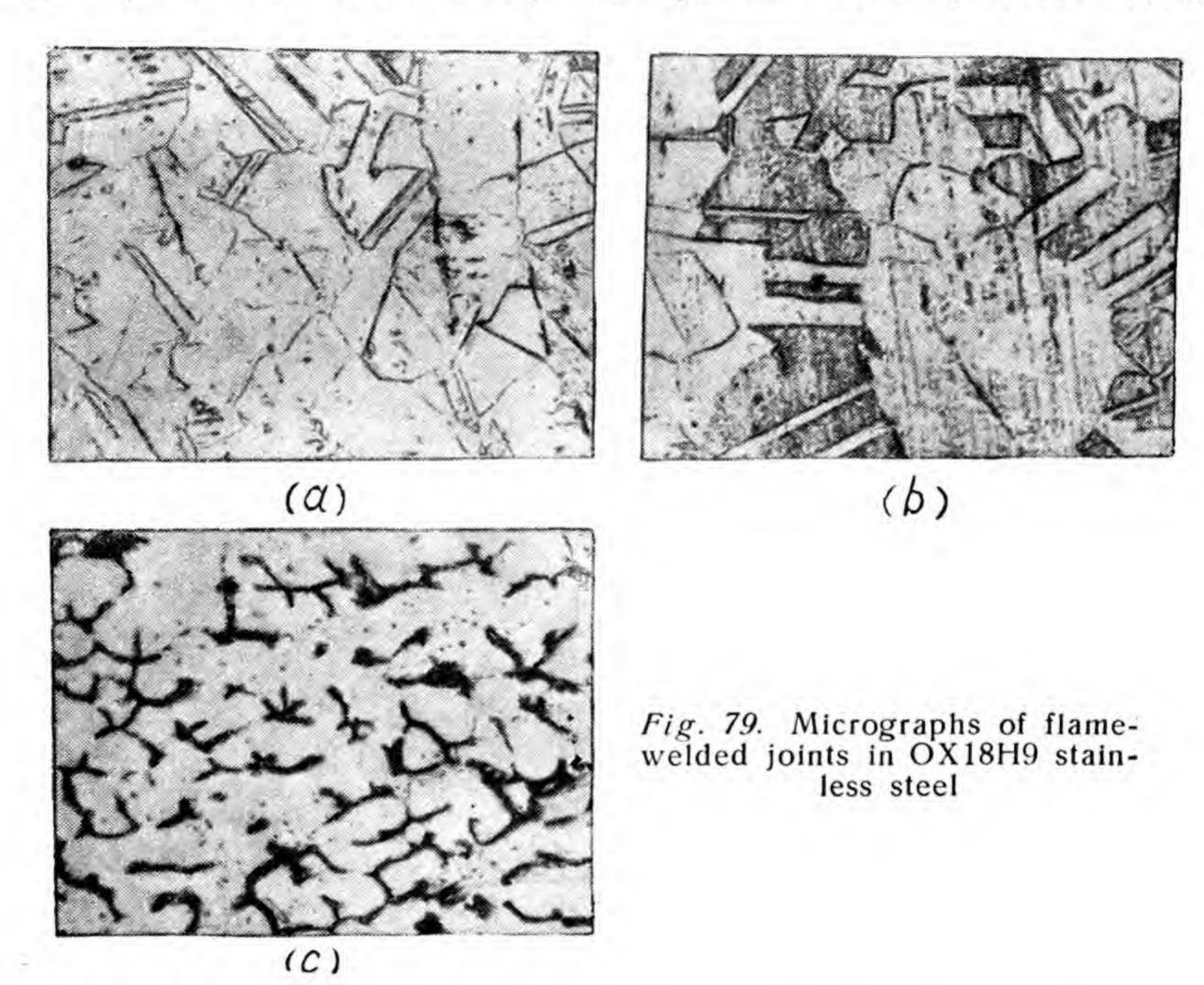
their growth, i. e., recrystallisation. In the case of quenched and tempered steels, the welding heat produces an additional high temper also impairing the strength of the steel.

The ageing area embraces the metal heated to 200-400° C and is observed in rimmed converter and Bessemer low-carbon steels with an increased gas content. Thermal ageing sharply reduces the impact strength of the metal.

It should be noted that the above delineation of the areas in the heat-affected zone is an arbitrary one. Indeed, they may overlap one another or be absent altogether, as the ageing area in killed steels.

Fig. 78a and b shows typical macrographs of welded joints in medium-carbon steel 1 and 9 mm thick. In both cases, the weld metal is dense, and the heat-affected zone is rather broad, measuring some 7 mm across in the 1-mm steel and about 15 mm in the 9-mm steel. The weld metal in both cases has a coarse-grained

microstructure (Fig. 78c for the 1-mm steel and Fig. 78d for the 9-mm steel). The fusion area between the weld metal and the parent metal shows coarse-grained ferrite-plus-pearlite structure due to appreciable overheating (Fig. 78e for the 1-mm steel and Fig. 78f for the 9-mm steel). The parent metal (medium-carbon

(a)

(b)

(c)

Fig. 79. Micrographs of flame-welded joints in OX18H9 stainless steel

steel) in both cases has a dotted close-grain ferrite-plus-pearlite structure (Fig. 78g).

Fig. 79 shows micrographs of the weld metal (at a), the fusion area (at b) and the parent metal (at c) of Grade OX18H9 stainless steel 2 mm thick.

The gas welding of stainless chrome-nickel steels rather heavy in thickness (over 5 mm) is characterised by the rejection of the chromium carbide along the boundaries of the austenite grains. This is brought about by the relatively low rate of cooling in the weld metal and the heat-affected area from 650° to 450° C.

In lighter sections (not over 2 or 3 mm), the welding operation goes on at a high speed, and the metal is overheated insignifi-

cantly. As a result, the rate of cooling in this temperature range is sufficiently high, and the rejection of the chromium carbide is retarded.
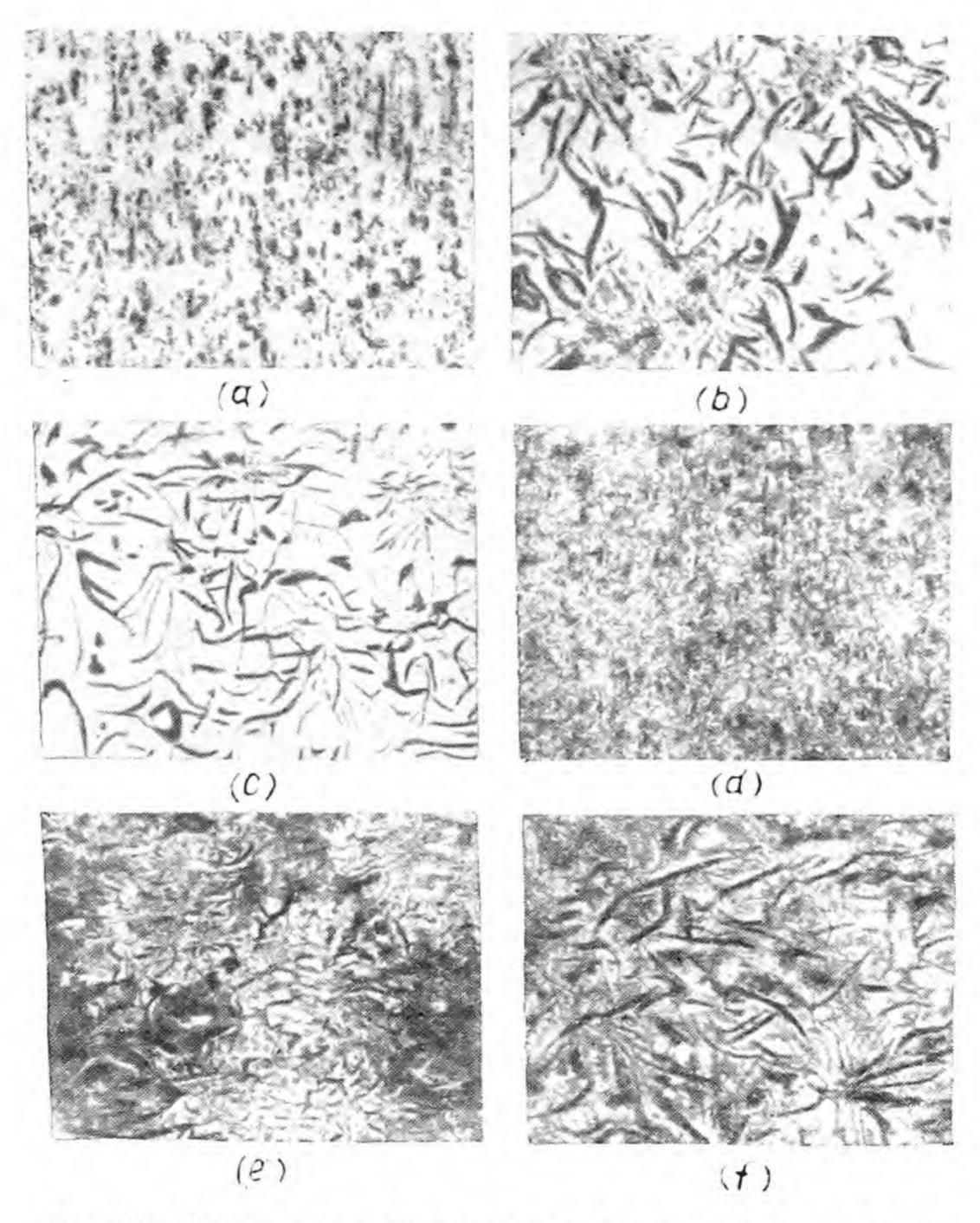
Fig. 80. Micrographs of flame-welded joints in grey iron 6 mm thick

The weld metal consists of solid-solution grains (austenite) with the carbide segregated along the grain boundaries (Fig. 79*a*), while the fusion area (Fig. 79*b*) and the parent metal (Fig. 79*c*) are austenitic duplex structures.

Grey cast iron is mostly welded by the oxy-acetylene process, as the torch can be conveniently used for concurrently heating the cast-iron workpiece to slow down the rate of cooling and facilitate graphitisation.

The gas welding of cast iron is characterised by the retention of its structure — free carbon in the form of flakes (Fig. 80).

(a)

(b)

(c)

Fig. 81. Micrographs of a flame-weld in copper 8 mm thick:
a — weld metal; b — fusion zone; c — parent metal
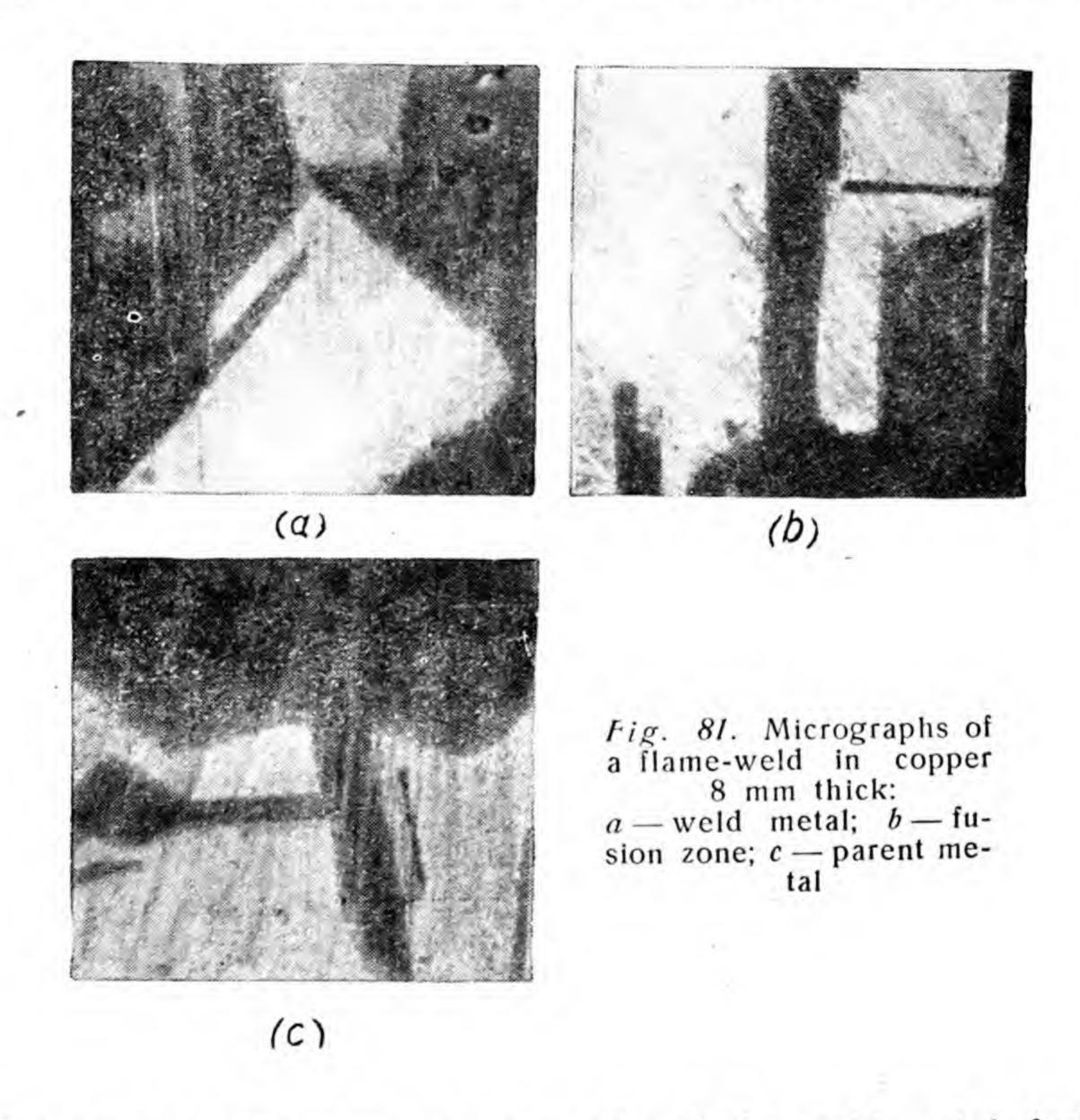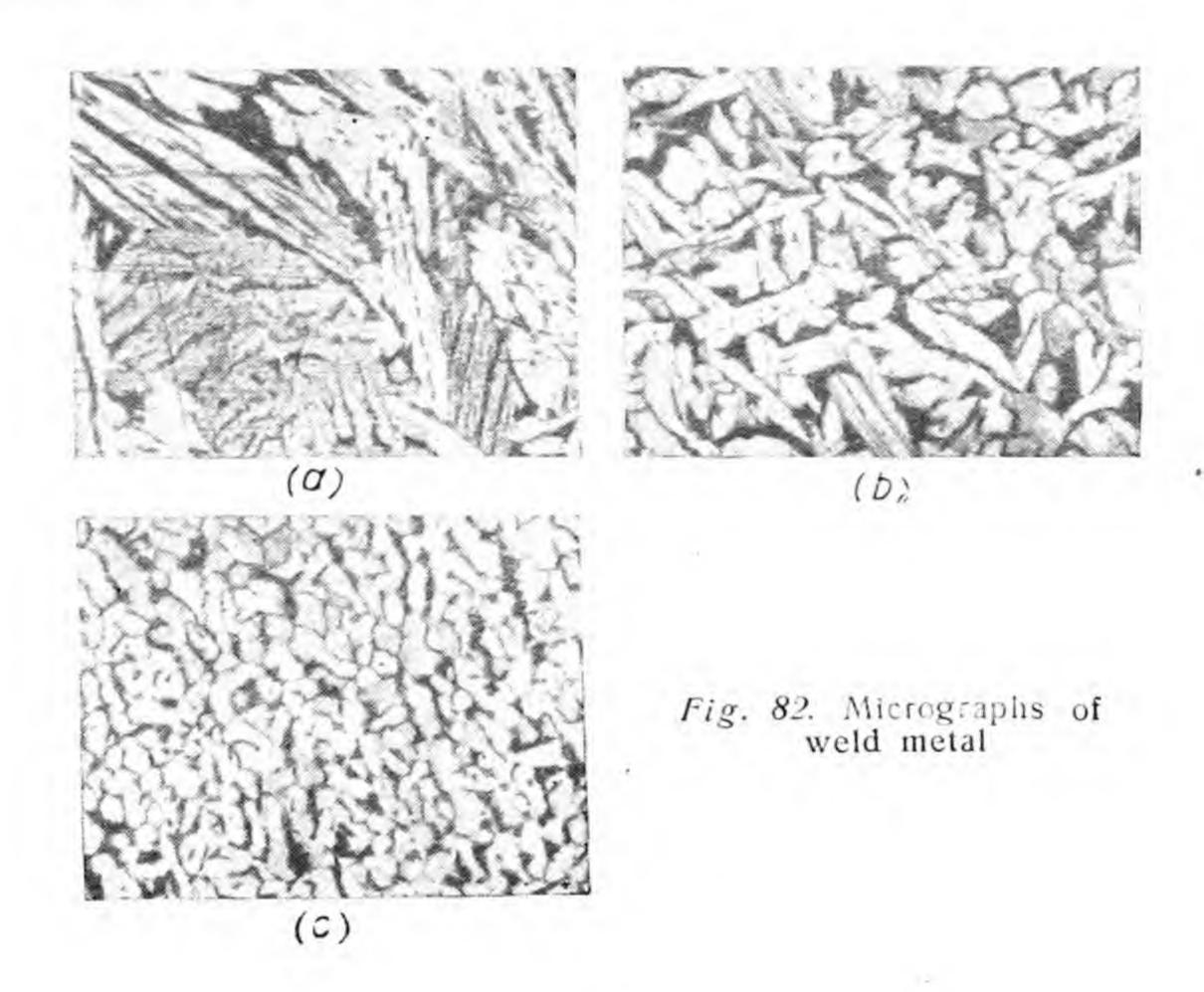
Fig. 80 a, b and c are micrographs of the weld metal, fusion area and parent metal in grey cast iron 6 mm thick prior to etching. Both the weld metal (at a) and the fusion area (at b) show graphitic inclusions of the dotted type in the weld and of the flake type in the fusion area. The same areas after etching are shown in Fig. 80 d, e and f. All the areas show a ferritic-pearlite-graphitic structure not affected by the welding heat.

When properly carried out, the gas welding of copper produces welded joints of sufficient mechanical properties, especially
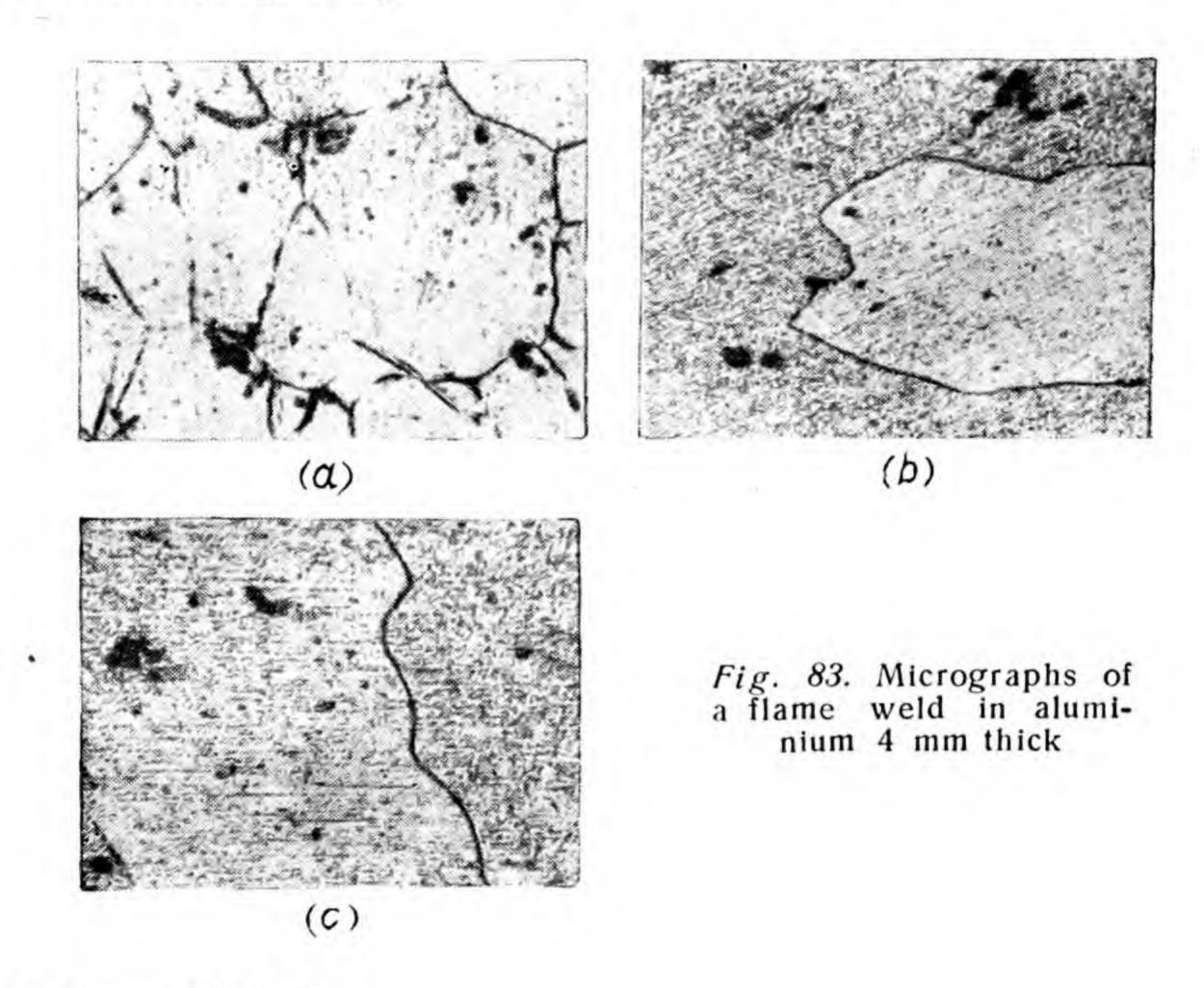
strength (annealed and quenched in water, copper has a tensile strength of 24 to 25 kg/mm² and an elongation of 40 to 50 per cent). Fig. 81 *a, b* and *c* shows, respectively, micrographs of the weld metal, fusion area and parent metal in copper. All of the areas have a coarse-grained, polyhedral structure. In addition, the parent metal displays duplex structures.

(a)

(b)

(c)

Fig. 82. Micrographs of weld metal

Compared with arc welding, the oxy-acetylene process as applied to brass reduces the vaporisation of zinc which has a vaporisation point close to the melting point of brass. In gas welding, too, especially when a powdered flux is employed, from 5 to 25 per cent of the zinc contained in the parent and filler metal may be vaporised, depending on the operator's skill and the welding procedure adopted. The weld metal shows the presence of two phases: the α-phase (light-coloured areas) and the β-phase (dark-coloured areas in Fig. 82 *a, b* and *c*). The grains in the weld metal and fusion area are larger than in the parent metal.

Gas welds in aluminium show a microstructure incorporating polyhedral grains which are finer in the parent metal (Fig. 83*c*)

and somewhat coarser in the fusion area (Fig. 83*b*) and in the weld metal (Fig. 83*a*).

(*a*)

(*b*)

(*c*)

Fig. 83. Micrographs of a flame weld in aluminium 4 mm thick

4. Welding Defects

The most important defects in gas welds are lack of fusion, undercuts, burnt metal, overheating, insufficient throat, porosity, slag inclusions and cracks.

Lack of fusion is the lack or even absence of union between the parent and weld metal. It may be caused by dirt on the edges, too small a gap between the fusion faces, improper joint preparation, a weak flame, or excessive rate of flame travel. Lack of fusion ranks among the most dangerous welding defects.

Undercuts appear at the toes of a weld (Fig. 84*a*). They may be caused by excessive flame output, too high a rate of fuel flow, nonuniform melting of filler metal, improper manipulation of the rod, etc. Undercuts reduce the effective cross-section of welds and are therefore very dangerous.

Burnt metal is produced when the flame is played on the puddle too long in the presence of excess oxides in the molten pool or filler metal or excess oxygen in the flame, or when the oxidising zone of the flame is allowed to come in contact with the metal

(which usually occurs when the inner cone is held far from the puddle surface).

Overheating results from the prolonged effect of the flame and is independent of the action of oxygen. It may be found both in the weld metal and in the heat-affected area. In gas welding, the overheated area may be fairly large.

(a)   *(b)*
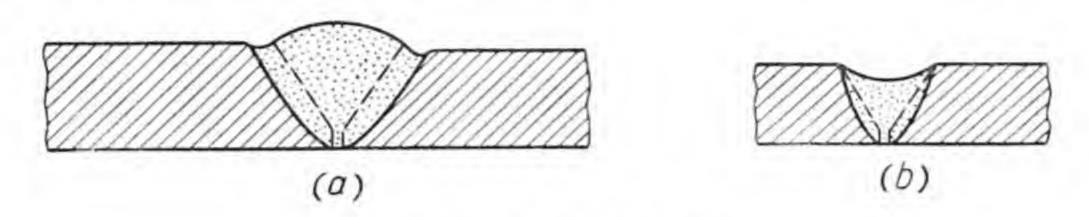
Fig. 84. Weld defects:
a — undercut; b — insufficient throat

Insufficient throat usually results from the insufficient amount of filler metal added to the weld pool (Fig. 84*b*). This weakens the weld and may lead to service failure.

Porosity is the presence of gas pockets produced by the gas evolution during the chemical reactions that take place in the molten pool and by some gases contained in the flame, especially hydrogen, dissolving in the molten metal. In gas welding it is a minor defect, as the weld metal solidifies slowly and a great proportion of the gases have time to escape.

Slag inclusions are due to the dirt left on the edges to be welded and filler metal, excess oxygen in the flame, poor puddling, high rate of solidification due to insufficient heating, and some other causes. Slag inclusions weaken the weld and are a serious defect.

Cracks are the most dangerous defect in welded joints. They may develop both in the weld metal and the heat-affected zone. The causes may be locked-up stresses due to nonuniform heating and structural changes in the heat-affected zone; embrittlement of the metal at temperatures close to the solidus; and the rigidity of the workpiece. For the most part cracks appear in rigid articles from hardenable steels in which the heat-affected zone possesses reduced plasticity and ductility. Locked-up stresses are especially great and often give rise to cracks in air-hardening structural steels which undergo volumetric changes in the near-weld area due to martensitic transformations.

CHAPTER XI

A GENERAL OUTLINE OF GAS WELDING

This chapter deals with the general procedures and techniques of gas welding. Though they are discussed for plain low-carbon steels, these procedures and techniques equally apply to all metals. Special problems of gas welding for alloy steels, cast iron and nonferrous metals are examined in the subsequent chapters.

1. Types of Welded Joints

The butt joint is the most commonly used joint in gas welding both for light sections and heavy gauges. On thicknesses up to 2 mm, preference should be given to the closed double-flanged butt joint (Fig. 85a) or the closed square butt joint (Fig. 85b), both made without filler metal. On thicknesses from 2 to 5 mm, the open square butt joint should be selected (Fig. 85c). For thicknesses upwards of 5 mm, joints with the joint preparation from one or both sides (Fig. 85 f and g) may be recommended.

The lap joint is undesirable and, sometimes (on metal thicknesses over 3 mm), even intolerable because of nonuniform and excessive local heating, increased locked-up stresses, deformation and cracks in the weld and parent metal, especially when the workpiece is sufficiently rigid. The lap joint may, however, be employed on thicknesses under 3 mm (Fig. 85h), such as in aircraft construction, where stiffening washers have to be welded between mating parts.

The Tee joint (Fig. 85i) is quite possible for light gauges up to 3 mm. This type of joint is widely employed to weld stiffeners in aircraft and other thin-walled structures. On thicknesses of over 3 mm, however, the Tee joint is undesirable as it entails appreciable local heating.

The corner joint (Fig. 85d) is suitable for both light and fairly heavy gauges. In some cases, the corner joint can be welded without any filler metal, the weld being formed by melting off the edges of the parent metal.

The edge joint (Fig. 85e) is rather commonly used in gas welding. As is the case with the closed double-flanged butt joint, it is usually welded without filler metal and lends itself readily to machine processes.
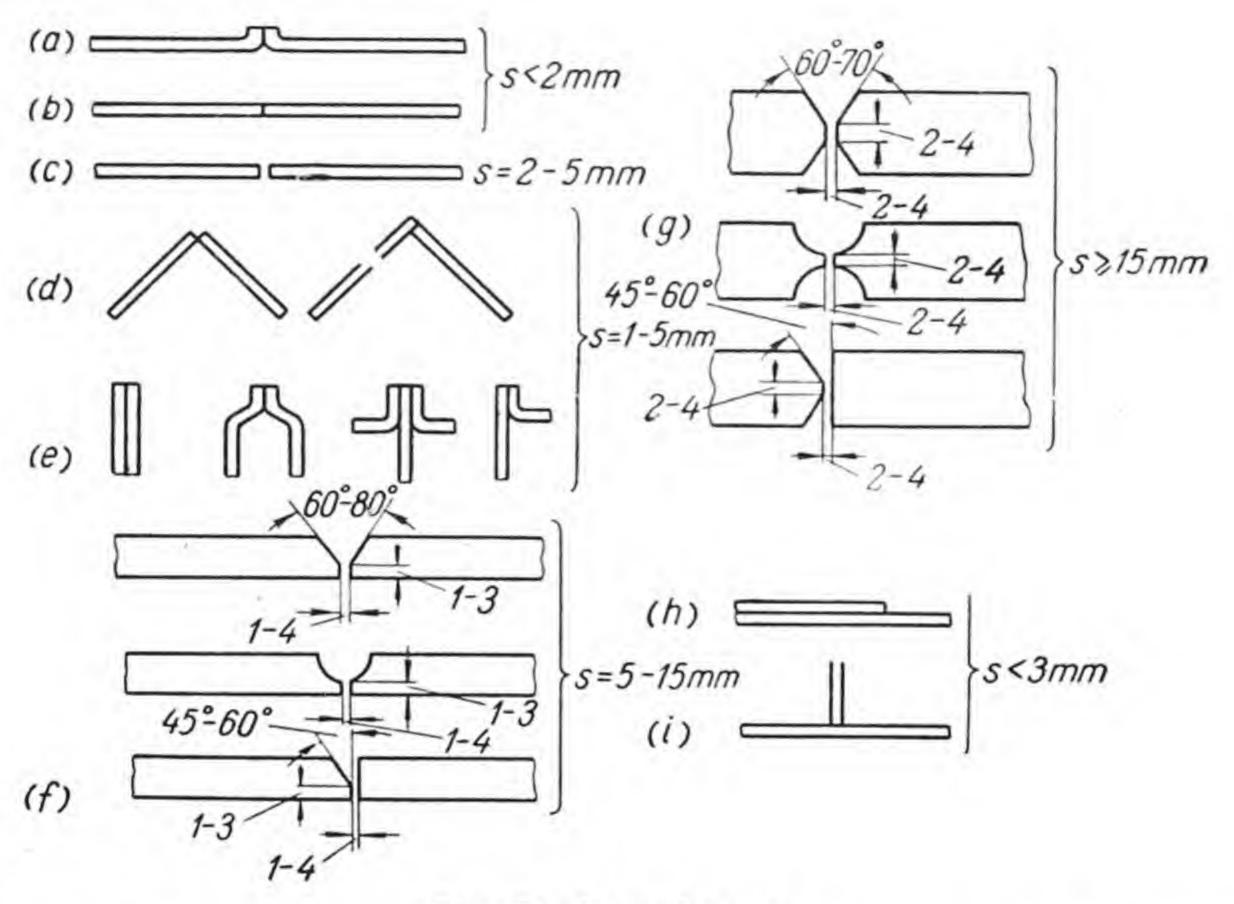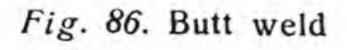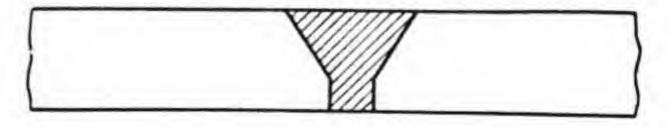
Fig. 85. Joint designs:
a, b, c, f, g — butt joints; *d* — corner joints; *e* — edge joints; *h* — lap joint; *i* — Tee joint

As distinct from the butt joints in which the weld metal is deposited in the groove between the edges of the parent metal (Fig. 86), the lap and Tee joints use what is known as the fillet

Fig. 86. Butt weld
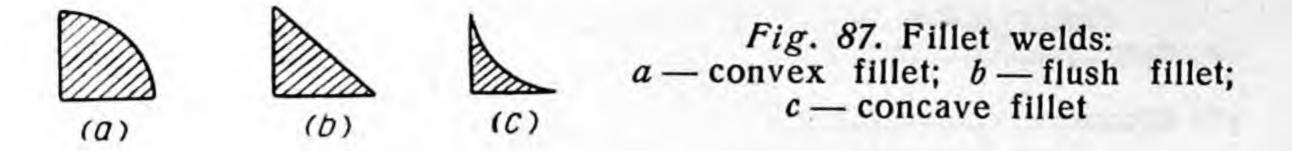
weld where the metal is deposited on the surface of the parts to be welded.

There are three types of fillet welds: convex, flash and concave (Fig. 87). Concave fillet welds are common in aircraft structures where it is essential to increase the fatigue strength and to reduce the weight of an article.

By their position, all welds are classed into: downhand (or flat) welds deposited in any direction on a horizontal surface so that the flame is above the face of the weld (Fig. 88a); vertical

Fig. 87. Fillet welds:
a — convex fillet; b — flush fillet;
c — concave fillet

welds deposited on a vertical surface in a vertical direction (upwards or downwards, as shown in Fig. 88b); inclined welds deposited up or down an inclined surface (Fig. 88c); horizontal welds * deposited on a vertical surface in a horizontal direction (from left to right or from right to left, as shown in Fig. 88d);
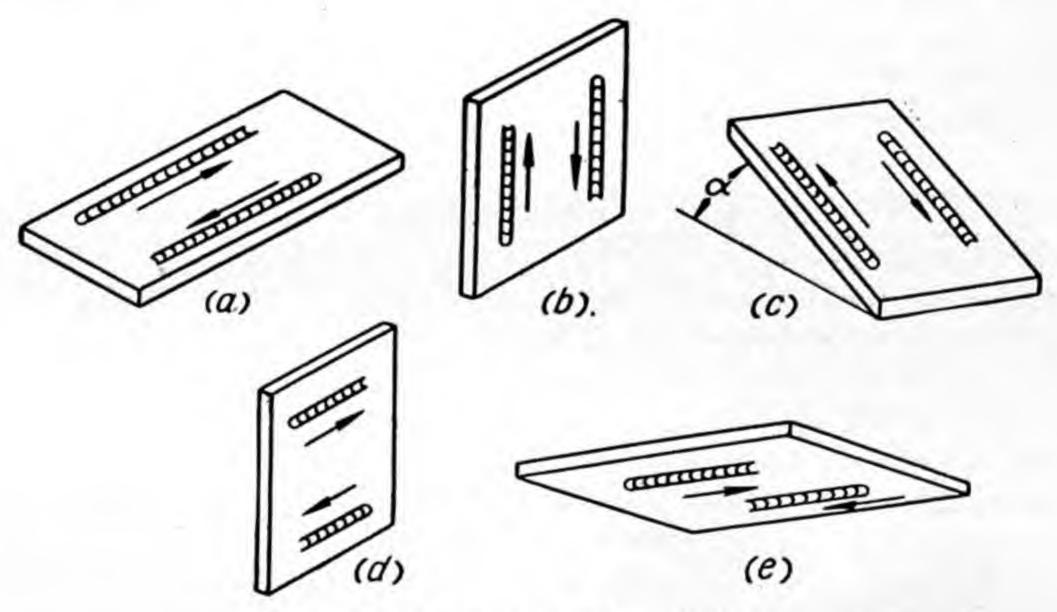
Fig. 88. Positions of welding

and overhead welds deposited on a horizontal surface in any direction so that the face of the weld is above the flame (Fig. 88e).

As regards the load acting on a weld, all welds are classed into longitudinally loaded (running parallel with the direction of load); transversely loaded (running at right angles to the direction of load); and obliquely loaded (running at an angle other than normal to the direction of load).

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* It will be noted that this classification differs from what is adopted for fillet welds in the U. S. and the U. K. See "American Welding Society Definitions", p. 14, and BS 499: 1952, p. 41—Tr.

2. Rightward (Backhand) and Leftward (Forehand) Welding

In gas welding, the direction of travel and the tilt of the torch have an appreciable effect on the speed and quality of welding. There are two techniques differing in torch travel and tilt — rightward (or backhand) welding and leftward (or forehand) welding.

In rightward (backhand) welding, the flame is directed back against the completed weld, and the weld progresses from left to right. The flame travels forward of the filler rod, and the weld is formed as shown in Fig. 89a.
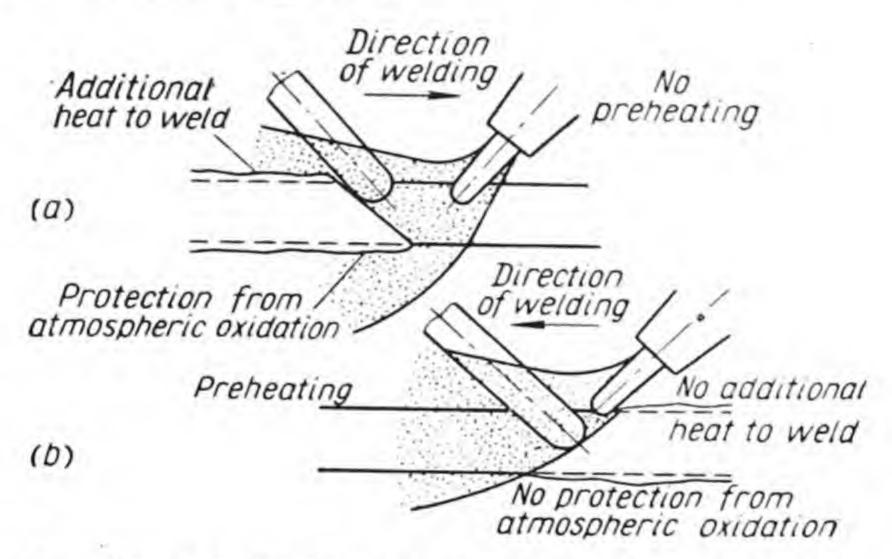
Fig. 89. (a) Rightward (backhand) and (b) leftward (forehand) techniques of welding

In leftward (forehand) welding, the flame is directed ahead of the completed weld, and the weld progresses from right to left. The flame travels back of the filler rod, and the weld is formed as shown in Fig. 89b.

In rightward welding, where the flame is directed towards the completed weld, the latter is better shielded from atmospheric oxidation and the rate of cooling is slowed down.

The appearance of the weld, especially on light sections, is better with leftward welding, however, as the operator can clearly see the solidifying front of the puddle and make the bead more uniform in height and width.

Experience has shown that the leftward technique is more efficient on thicknesses up to 3 mm, while the rightward technique yields better results on heavier gauges (especially on groove joints).

M. Kh. Shorshorov has investigated the efficiency and rate of heating and heat penetration with both rightward and left-

ward welding as a function of metal thickness (see Sec. 4, Chapter VIII). According to him, the higher rate of heating obtained with rightward welding on thicknesses upwards of 5 mm with joint preparation is mainly due to the shorter distance between the inner cone of the flame and the weld pool and also to the fact that the fusion faces and the weld deform the flame differently than in leftward welding.

It has also been found that the higher rate of welding obtained with the leftward technique on light gauges (under 3 mm) on closed or square joints is due to the preheating action of the flame streamer while the distance between the inner cone and the metal is the same as in rightward welding.

This is, however, true of light gauges, as on heavier gauges the heat of the flame envelope proves insufficient to preheat the metal and thus raise the rate of heating and welding.

Investigations into the welding of fairly heavy gauges of steel (5 to 12 mm) by the rightward and leftward techniques by N. N. Klebanov have proved the following.

1. The rate of acetylene flow for the rightward technique should be kept at 150 lit/hr per millimetre of metal thickness, as against 100 to 120 lit/hr for the leftward technique.

2. The edges in rightward welding should be bevelled by approximately 10 to 15° less than for the leftward technique; the torch should be advanced along the joint without weaving, while the welding rod should be weaved for the better puddling of the weld pool and for a better weld appearance.

3. The angle between the joint and the torch taken in a longitudinal plane should be by 10 to 20° greater in rightward than in leftward welding.

4. Under the above conditions, the welding speed is 20 to 25 per cent higher and the fuel gas consumption 15 to 25 per cent lower with rightward welding.

5. The economic efficiency of the rightward technique is better on heavier gauges. The reduction in welding time and gas consumption obtained with rightward welding is due to the smaller bevel angles with which less filler metal has to be deposited, and to the better utilisation of the flame heat.

6. The heat-affected zone is smaller with rightward than leftward welding. The Widmanstätten structure occurring in the gas welding of heavy-gauge steel, develops, however, with rightward technique as well.

7. Rightward welding provides better shielding against atmospheric oxidation of the weld metal and slows down its cooling, for which reasons the weld metal is denser, stronger and tougher.
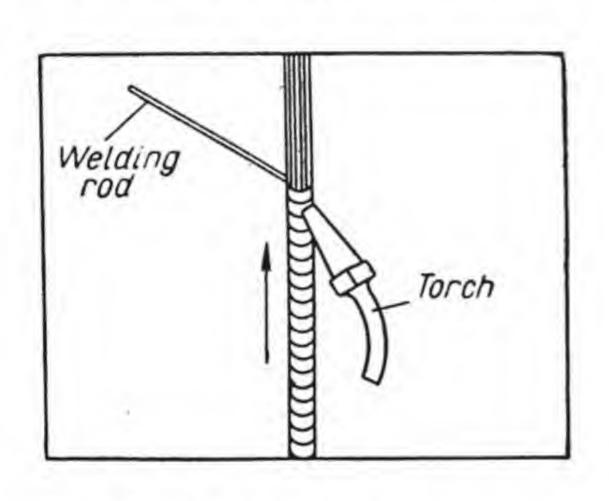
3. The Effect of Welding Positions on the Weld Quality
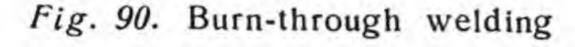
The formation of a gas weld is to a great extent dependent on (1) the pressure and direction of the gas flame; (2) the motion of the welding rod end; (3) the weight of the drop of molten metal; and (4) the surface tension of the molten metal.

In downhand welding the effect of the last two factors is insignificant.

With the fuel mixture issuing from the nozzle at 120 to 150 m/sec, the pressure of the flame is about 100 to 120 g/cm^2. This is sufficient for the molten pool to be thoroughly puddled and forced towards the completed weld. The puddling action of the rod also adds to this effect.

Fig. 90. Burn-through welding
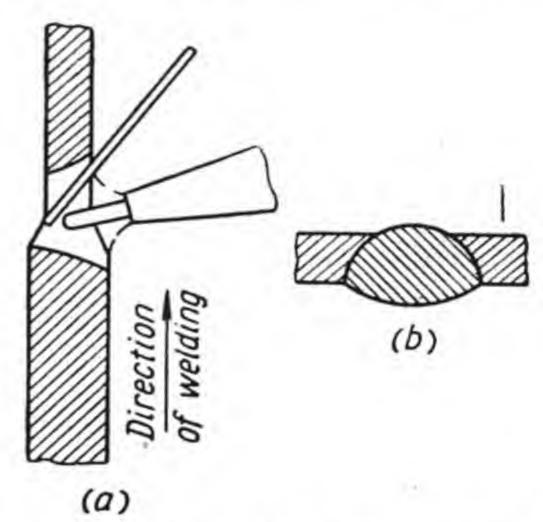
Fig. 91. Upward welding of a vertical joint

In vertical and overhead welding, the pressure of the flame, the action of the welding rod, and the surface tension of the weld pool tend to retain the metal in the pool, while the force of gravity tends to cause the drop of molten metal to break away from and run out of the puddle.

Whether rightward or leftward welding is preferable in a given case depends on which position will be used. In the downhand position, both techniques are equally efficient, and the welding speed is solely governed by the metal thickness on hand. In the case of vertical welds made upward, the leftward technique seems more convenient with the torch and rod positioned as shown in Fig. 90. Sometimes, upward vertical welds can be welded by the righthand technique, which may be called burn-through welding, which is applicable to vertical closed butt joints in steel plate 8 mm thick. The operator starts the weld by burning through the plate, fills it up with filler metal which is

deposited on the lower edge of the hole, expands the hole in the upward direction, and gradually moves the torch up along the joint (Fig. 91*a*). Thus, the bead is formed on both sides of the joint (Fig. 91*b*) and is more dense than a downhand weld.

Horizontal welds are better made by the rightward technique where the flame is directed back against the completed weld and
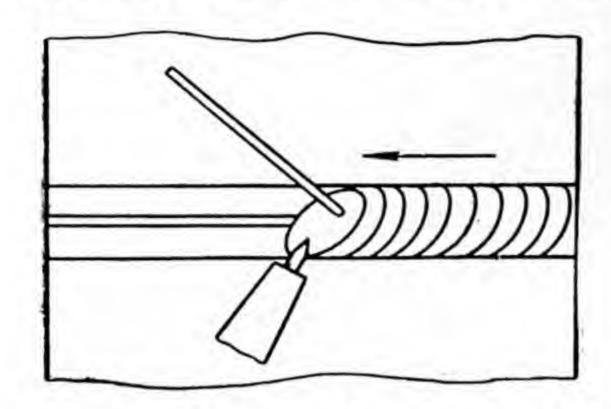
Fig. 92. Welding of a horizontal joint

prevents the molten metal from running out of the pool. As distinct from the conventional rightward technique, the one applied to horizontal welds is carried out from right to left, and the puddle is maintained askew in order to facilitate the formation of the bead (Fig. 92).

(a)   (b)
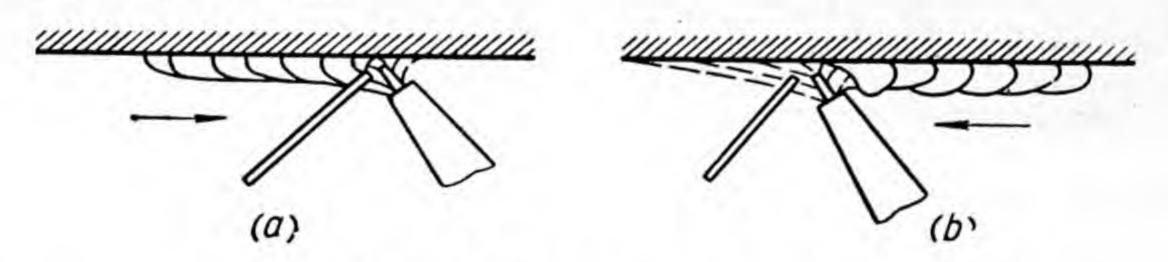
Fig. 93. Overhead welding:
a — rightward technique; *b* — leftward technique

Overhead welds are made easier by the righthand technique, as in this case both the rod end and the pressure of the flame prevent the molten metal from running out of the puddle (Fig. 93*a*). With the leftward technique, on the other hand, there are overlaps and the appearance of the weld is unsatisfactory (Fig. 93*b*).

4. Preparation of the Workpiece for Welding

(1) **Cleaning of the Edges.** Prior to welding, the edges of the workpiece and the adjoining area (10 to 20 mm wide on either side) should be thoroughly cleaned of mill scale, rust, paint, grease and other contaminants. This can be done by the torch flame and is called flame blasting. As steel and scale have dif-

ferent thermal coefficients of expansion, the scale, when heated, will easily come off the workpiece surface. Paint and grease are simply burned by the flame. After flame blasting, the surface of the workpiece should be wiped with a steel-wire brush to metallic lustre.

The surface of critical jobs small in size should be either pickled or sand-blasted.

The cleaning operation should be given proper attention, as the dirt on the workpiece will inevitably result in poor fusion and nonmetallic inclusions in the weld metal.

(2) Tack-welding. Prior to welding, the parts to be welded together are tacked to hold them fast in place and to maintain the desired gap and relative position of the parts. Tacking is done by means of short tack welds. Their length and spacing
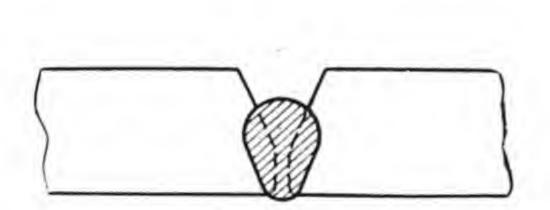
Fig. 94. A cross-section through a tack weld
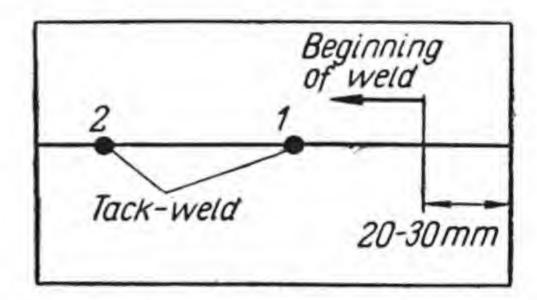
Fig. 95. Location of tack welds on short joints

vary with the metal thickness and the length of the joint. On small articles from light-gauge steel tack welds are not more than 5 mm long and are spaced at 50 to 100 mm. When long joints are to be welded in heavy plate tack welds may be 20 to 30 mm long and spaced 300 to 500 mm apart.

The cross-section of tack welds also depends on the thickness of the workpiece, increasing with the latter. In joints with bevelled, veed, or otherwise prepared edges, tack welds do not fill the whole of the groove space (see Fig. 94).

In tacking, as much care must be exercised as in the actual welding, operating under the same conditions. Poor penetration or fusion in tack welds will result in poor welded joints also suffering from lack of fusion.

Another important feature in tacking is the sequence in which tack welds are to be applied. This sequence varies with metal thickness and the length of the joint. On light gauges and short joints tack welds may be applied as shown in Fig. 95. On long joints, the sequence shown in Fig. 96 should be preferred.

Sometimes skilled operators weld butt joints without tacking. Instead, the plates are placed at an angle to, rather than parallel with, one another. As the welding progresses, the transverse contraction causes the plates to draw together, and the desired gap is thus maintained constant along the entire length of the joint.

(a)   (b)
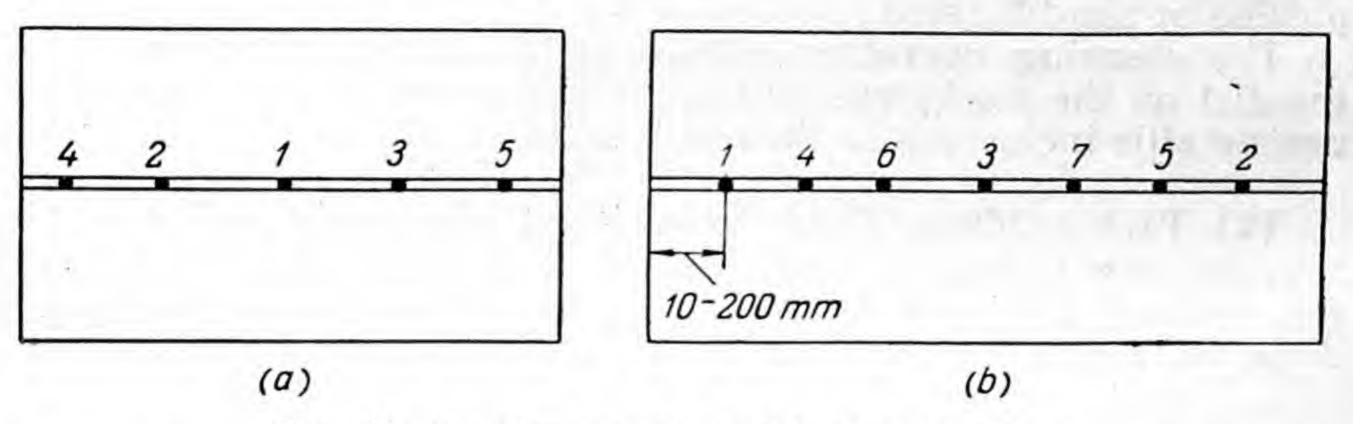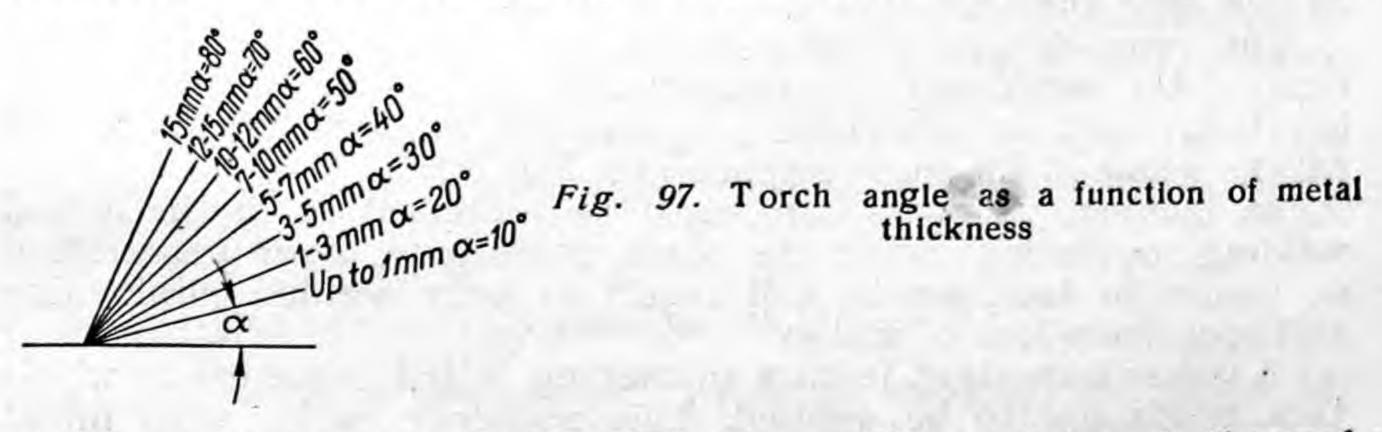
Fig. 96. Location of tack welds on long joints:
a — from plate centre outwards; b — from plate edge inwards; *1-5* and *1-7* sequence of tack-welding

This purpose can also be served by a wedge kept in the seam and moved along ahead of the torch as the weld progresses.

5. Conditions of Gas Welding

The main variables of gas welding, in addition to welding speed which may be any in the manual process, are the torch angle, heat input, and welding rod diameter.

(1) The Torch Angle. This is the angle between the joint and the torch tip taken in a longitudinal plane. It increases with the

Fig. 97. Torch angle as a function of metal thickness

thickness of the metal (Fig. 97), its melting point and thermal conductivity. For copper, the torch angle is 60 to 80°, and for lead usually under 10°.

Furthermore, the torch angle can vary in the process of welding. In the beginning, when more heat is necessary to raise the temperature of the workpiece and obtain the pool of molten metal quicker, the torch angle is usually the greatest (80 to 90°). At the end of the joint, in order to avoid burning through and to fill up the crater, it is reduced to a minimum so that
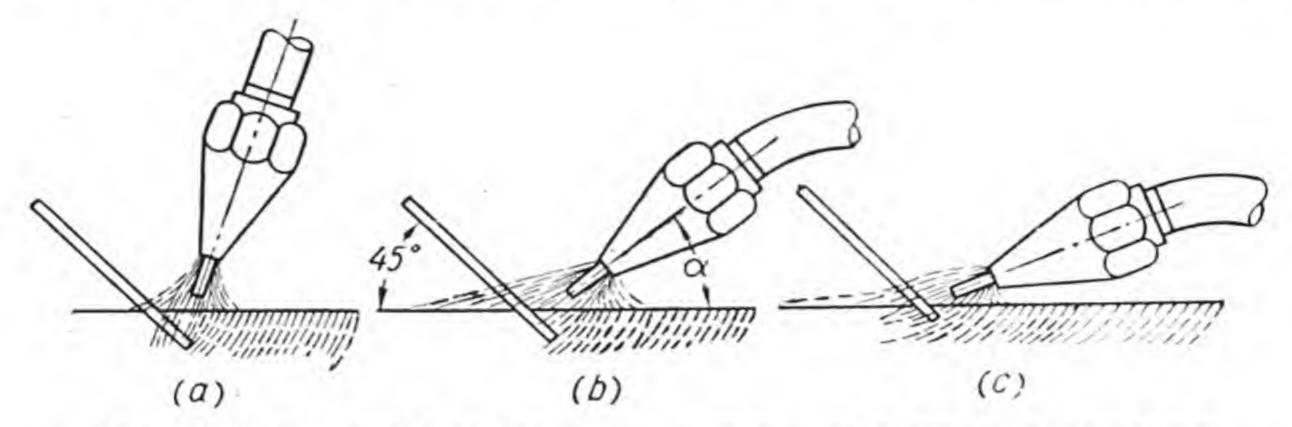
Fig. 98. Torch angle in the various stages of a welding operation: *a* — preheating prior the welding; *b* — welding; *c* — end of welding (filling up the crater)

the flame almost skims over the metal (Fig. 98). In between, the torch angle is maintained according to the thickness and nature of the metal.

(2) Heat Input. Heat input is usually expressed by the hourly rate of fuel gas flow. Heat input should be increased as the thickness, melting point and thermal conductivity of the metal increase. For low-carbon and low-alloy steels, the hourly rate of acetylene flow can be found by the following empirical equations:

for leftward welding:

$$V_{C_2H_2}(100 \text{ to } 120)S \text{ lit/hr};\qquad\qquad\text{(XI.1)}$$

for rightward welding:

$$V_{C_2H_2} = (120 \text{ to } 150)S \text{ lit/hr},\qquad\qquad\text{(XI.2)}$$

where S is the thickness of the steel to be welded in mm.

For cast iron, brass, bronze and aluminium alloys, about the same heat input is required as for steel.

For copper, which has a high thermal conductivity and a rather high melting point, heat input can be calculated by the following equation:

$$V_{C_2H_2} = (150 \text{ to } 200)S \text{ lit/hr}\qquad\qquad\text{(XI.3)}$$

if a single torch is employed; and by the equation:

$$V_{C_2H_2} = (120 \text{ to } 150)\, S \text{ lit/hr} \qquad (XI.4)$$

if two torches are used. The heat input from the preheating torch can be then found by Eq. (XI.3).

In the course of welding the tip expands and, as a result, more oxygen is added to the fuel mixture, which fact often leads to the oxidation of the weld pool. To counteract this, the desired heat input should initially be obtained with the acetylene valve only partially opened. Then the operator will be able to compensate for the oxygen added by the expansion of the tip by opening the acetylene valve. In practice, this is sometimes accomplished by using a tip one size larger than is necessary for the heat input calculated by Eq. XI. 1 through XI. 4.

(3) Welding Rod Diameter. According to V. G. Nauman, the welding rod diameter to be used depends on the welding technique adopted. For leftward welding, the welding rod should be greater in diameter than for rightward welding. For steel 12 to 15 mm thick the diameter of the welding rod or wire can be found by the following empirical equations:

for leftward welding:

$$d = \frac{s}{2} + 1 \text{ mm}; \qquad (XI.5)$$

for rightward welding:

$$d = \frac{s}{2} \text{ mm}, \qquad (XI\ 6)$$

where d is the rod diameter in mm and s is the steel thickness in mm.

For steel upwards of 15 mm in thickness, the welding rod should be 6 to 8 mm in diameter.

6. Motions of the Torch and Welding Rod

The motions of the torch and welding rod have a direct bearing on the quality of the weld. These motions vary with the welding position, metal thickness and weld size.

Fig. 99 shows them for the rightward welding of closed joints in plate over 3 mm thick and for the leftward welding of both open and closed joints in the heavier metal, in the downhand position. As will be seen, the motion of the welding rod is the opposite of the torch's. This ensures proper penetration, fusion and weld appearance. For fillet welds, a motion similar to the dia-

gram in Fig. 100 should be used. There should be a hesitation in the motion for a short interval at the fusion faces.
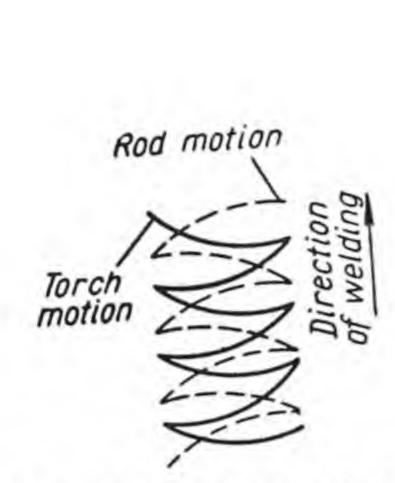
Fig. 99. Motions of the torch and welding rod on metal thickness over 3 mm
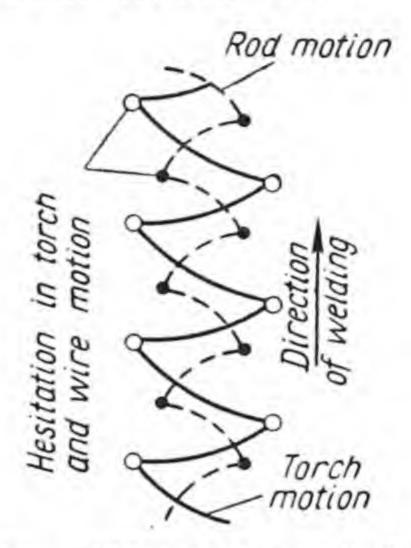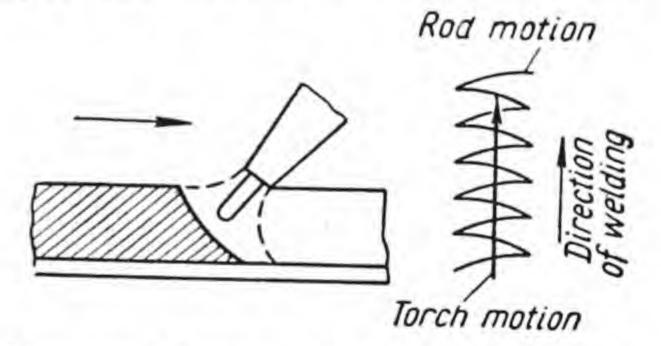
Fig. 100. Motions of the torch and welding rod for normal fillet welds

When welding plate over 5 mm in thickness by the rightward technique, the flame is moved deeper into the groove (Fig. 101)

Fig. 101. Motions of the torch and welding rod in welding grooved joints by the rightward technique

and advanced without weaving. The puddling action is supplied by the end of the welding rod.

3rd increment	1st increment	4th increment	2nd increment	
9	3	12	6	3rd layer
8	2	11	5	2nd layer
7	1	10	4	1st layer

Direction of welding
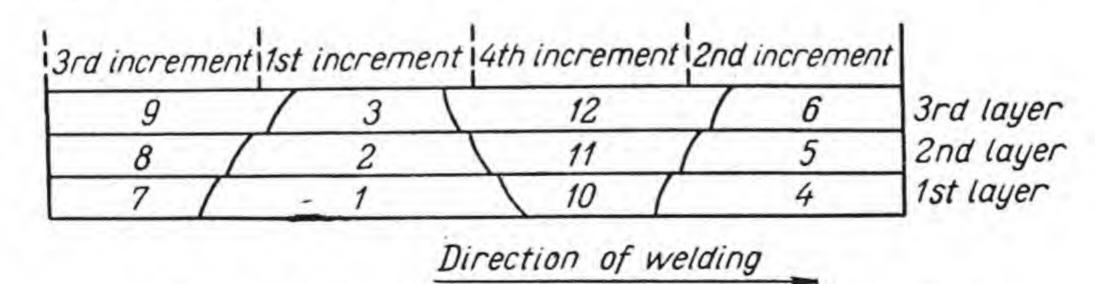
Fig. 102. Sequence of welding in multipas welding

Heavy-gauge steel plate is gas-welded but seldom because of low speed and the danger of overheating. In such cases, joints may be welded in one or several passes. The relevant welding sequence is shown in Fig. 102.

Steel sections less than 3 mm thick use flange welds welded without filler metal, the torch being given a spiral or a weave motion, as shown in Fig. 103.
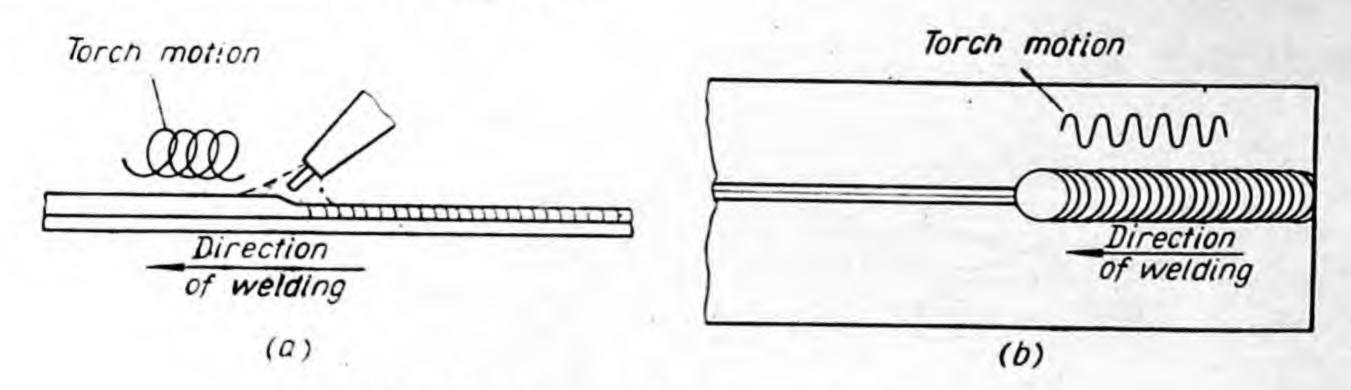
Fig. 103. Motion of the torch for flange welds:
a — spiral motion; *b* — weaving motion

Butt joints in light-gauge steel can advantageously be welded with filler metal by what may be called the fish-scale technique. As soon as a small puddle (4 or 5 mm in diameter) is formed, the end of the welding rod is dipped into it, until some of the rod is melted off, after which the rod is withdrawn from the puddle into the intermediate flame zone, the torch is brought closer to the metal surface in a snap circulator motion, and the next puddle is formed. Each next puddle overlaps the previous one by a third of its diameter (Fig. 104). If a high-quality weld
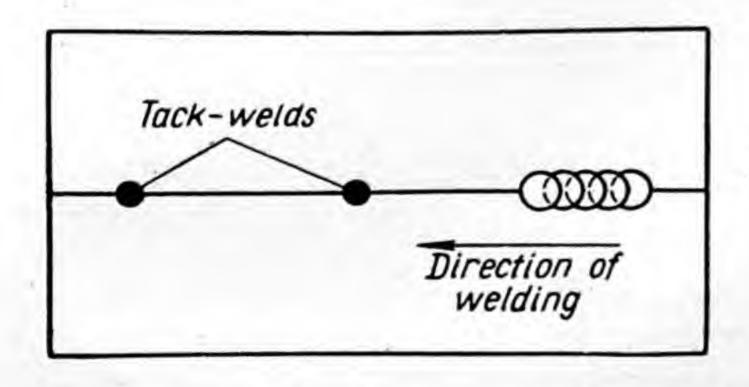
Fig. 104. The fish-scale technique of welding

is to be made by this technique, it is essential that (1) the end of the welding rod should not be removed from the intermediate flame zone or it will be oxidised and (2) the inner cone should be brought closer to the metal surface without touching it, or the weld metal will be carburised. The welds obtained by this technique are of high quality. In fact, fish-scale welding is the most commonly used technique in the aircraft industry for obtaining concave welds in low-carbon and low-alloy steels of low thickness.

CHAPTER XII

WELDING OF STRUCTURAL CARBON
AND ALLOY STEELS

At the present time, structural carbon and alloy steels are mainly welded by the manual, semiautomatic and automatic submerged arc processes using heavy-coated and consumable electrodes and by the inert-gas and CO_2-shielded arc processes. Still, gas welding is continued to be employed in erection, repair and batch production of light-gauge articles. Sometimes, gas welding is more advantageous as it is simple to carry out and uses simple equipment. This is especially important where the range of products fabricated is subject to frequent changes and the cost of installing new equipment and fixtures every time would be prohibitively high.

For these reasons our discussion will be limited to the basic procedures and techniques of gas welding as applied to structural and alloy steels.

Low-carbon steels (with not over 0.3 per cent carbon) lend themselves readily to welding. Low-alloy steels of the pearlitic class weld well with a carbon content of up to 0.35 per cent. A preheat to 200 or 300° is usually required to prevent cooling cracks in the weld metal and the heat-affected zone.

Martensitic steels, which possess poor weldability, should be preheated to 400 or 500° C and, sometimes, heat-treated (annealed or tempered) after welding. Austenitic steels have good weldability when their carbon content is under 0.2 per cent. Carbidic steels weld poorly and may be gas-welded only in exceptional cases.

1. Carbon Steels

In carbon steels containing over 0.3 per cent carbon, hot cracks may develop along the grain boundaries in the weld metal and in the near-weld area. These cracks usually appear at temperatures close to the solidus and are caused by the locked-up

stresses developing due to the constrained contraction of the metal.

Cooling cracks are brought about by rapid cooling and usually appear in the final stage of the martensitic transformations (below 200-400° C or after welding) due to the local thermal stresses and by the decomposition of the austenite into martensite, usually accompanied by changes in volume. The susceptibility of steels to such cracks increases with the carbon content. Cooling cracks always develop along the grain boundaries, but may propagate into the grains as well.

The thermal cycle in welding has an appreciable effect on cooling cracking. A prolonged holding at over Ac_3 gives rise to grain growth and embrittlement. Rapid cooling facilitates the formation of martensite in the weld and parent metal. Contrariwise, when the rate of cooling is low, the austenite decomposes into more stable structures, which fact ensures greater ductility and prevents cracking due to the internal stresses produced by structural changes in the steel.

Gas welding makes it possible to vary the rates of heating and cooling within broad limits and thus to control the strength and quality of the resultant weld.

The welding rod for medium-carbon (0.3-0.5 per cent carbon) and high-carbon (over 0.5 per cent C) steels should be low in carbon (Grades Cв-08, Cв-15 or Cв-15Г), as with more carbon the latter burns out and leaves a porous weld. The high mechaical properties of these steels may be retained by using welding wire alloyed with 0.5-1 per cent chromium and 2-4 per cent nickel and having the usual amount of manganese (0.5-0.8 per cent). The flame should be normal, with the rate of acetylene flow set at 75 lit/hour per mm of metal thickness. A higher specific rate of acetylene flow is fraught with overheating. The post-welding heat-treatment should include annealing or high temper at 750—800° with cooling in the air. It is sound practice to preheat the workpiece to 250 or 350°. The preheat reduces the rate of cooling and equalises the temperature gradient, for which reason internal stresses are reduced and hot and cooling cracks are less likely to occur. Thicknesses under 3 mm do not need preheating.

2. Low-alloy Steels of the Pearlitic Class

(1) Low-alloy structural steels, such as chrome-silicon-nickel-copper steel 15ХСНД (СХЛ-1 and НЛ-2), have increased strength, weld well and resist atmospheric corrosion efficiently. They are employed in engineering structures and chemical appa-

ratus. Though they are mainly arc-welded, gas welding is also used sometimes.

Steel 15ХСНД contains 0.12-0.18 per cent C, 0.4-0.7 per cent Mn, 0.4-0.7 per cent Si; 0.2-0.4 per cent Cu, 0.6-0.9 per cent Cr, 0.3-0.6 per cent Ni, and up to 0.04 per cent each P and S.

It has an ultimate tensile strength of 52 kg/mm², an elongation of 18 per cent and an impact strength of 8 kg-m/cm².

These steels should be welded with the normal flame and the heat input adjusted on the basis of Eqs. XI.1 and XI.2. Filler metal may be either low-carbon wire Св-08 and Св-15 or wire Св-08Г and Св-12М. Both leftward and rightward welding may be employed, depending on the metal thickness on hand and the welding position selected. To make the weld metal denser, it should be peened at a light-red heat (800-850°). If the peening is not to affect the ductility of the welded joint, it should be followed by normalising either in a furnace or by a torch and by cooling in the air. No flux is required.

(2) Molybdenum and chrome-molybdenum steels go to make tubing for high-pressure boilers. Low-alloy molybdenum steels (12M, 15M and 20M) and chrome-molybdenum steels (12XM, 20XM and 30XM) are heat-resistant grades containing 0.12-0.35 per cent C, 0.8-1.1 per cent Cr, and 0.15-0.65 per cent Mo. Steel 12XM contains up to 0.16 per cent C, 0.4-0.7 per cent Mn, 0.17-0.37 per cent Si, 0.4-0.6 per cent Mo, 0.8-1.1 per cent Cr, up to 0.3 per cent Ni, and not over 0.04 per cent of each S and P.

Molybdenum improves the tensile strength of the steel without impairing its ductility. At the same time, molybdenum improves its strength at elevated temperatures and decreases its tendency to creep. Furthermore, molybdenum contributes to deep hardening of steel, makes it fine-grained, and prevents temper embrittlement of low-chrome and other alloy steels. At the same time, molybdenum is a carbide-forming element and is mainly present therefore as MoC_2 or Fe_3Mo_3C, though some molybdenum dissolves in the ferrite.

Chromium raises the creep limit and heat resistance of molybdenum steels and forms refractory and dense oxides on their surfaces.

Gas welding is resorted to when tubes from these steels are welded in the fabrication and erection of boilers. In tube welding the rods should be of the grades listed in Table 23.

Steel 15XM should preferably be welded with Св-18XMA wire with a composition close to that of chrome-molybdenum steel 20XMA.

Table 23

Welding Wire for Molybdenum and Chrome-molybdenum Steels

Grade designation	Chemical composition, per cent							
	C	Mn	Si	Cr	Ni	Mo	S	P
Св-18ХМА	0.15—0.22	0.4—0.7	0.15—0.35	0.8—1.1	max. 0.3	0.15—0.25	max. 0.025	max. 0.03
Св-12ХМ	max. 0.12	0.4—0.7	0.15—0.35	0.8—1.1	max. 0.3	0.4—0.6	max. 0.03	max. 0.03
Св-12МХ	max. 0.12	0.4—0.7	0.15—0.35	0.45—0.65	max. 0.03	0.4—0.6	max. 0.03	max. 0.03
Св-12М	max. 0.12	0.4—0.7	0.15—0.35	max. 0.2	max. 0.3	0.4—0.6	max. 0.03	max. 0.03
Св-X5М	max. 0.12	0.4—0.7	0.15—0.35	4—6	max. 0.3	0.4—0.6	max. 0.03	max. 0.03

The ends of adjoining pipes should be machine-bevelled to an angle of $45 \pm 2.5°$ and have a root face of 0.5-0.7 mm and a gap of not over 0.5 mm.

Both the edge preparation and fit-up should be checked with a template, and the joints should be welded in a line-up jig. Each joint should be tack-welded at three points equally spaced round the circumference.

The flame should be normal, as the excess of oxygen produces refractory oxides of chromium and molybdenum.

Welding conditions

Pipe wall thickness, mm . .	up to 3	4-6	7-10
Rate of acetylene flow, lit/hr	300	500	750-1,200
Welding rod diameter, mm . .	3-4	4-5	4-6

When the ambient temperature is below zero, the joint should be uniformly preheated all the way round the circumference to 250 or 300° C with the torch.

The joint should be welded in increments. On wall thickness up to 5 mm the weld should be deposited in a single pass, while on thicknesses over 5 mm, in two passes. If complete penetration is to be obtained, it is good practice to melt the root faces of an increment thoroughly without adding any filler metal, then fill the groove with molten metal the whole depth and build up reinforcement over the length of the increment. In two-pass welding, the first layer should be deposited across the entire depth

of the pipe wall, and the second layer should serve as reinforcement. The same procedure is followed in welding the subsequent increments. The length of an increment should be such as to make the melting of the root faces feasible. Usually, it is 15 to 25 mm, for on greater lengths a shrinkage crack may develop in the metal, which will persist in the welded joint. Both rightward and leftward welding is applicable. Overheating must be guarded against, and the weld pool should be kept pasty so as to reduce the burning out of the molybdenum and chromium.

When the welding operation has to be interrupted, or upon completion of a weld, the torch should be turned away from the weld pool gradually so that the gases may escape freely and no surface pores form. The welding operation should never be interrupted until the joint has been completed at least half the way round the seam. When picking up an interrupted weld, the whole joint should first be heated to 250 or 300° C so as to prevent cracking.

Pipes are often welded by the fish-scale technique. Throughout the welding operation it is essential to see that the reinforcement is symmetrical with respect to the centre plane of the weld, and the weld metal blends smoothly with the parent metal. The width of the reinforcement should be approximately three times the wall thickness, and the height of the reinforcement, half the wall thickness.

After welding, it is a good plan to give welded pipes a local heat treatment to relieve locked-up stresses, to refine the structure and to improve the mechanical properties of the welded joints. For this purpose, the welded pipe should be uniformly heated with one or several torches over approximately five widths of the welded joint. Watch on the heating temperature may be kept by means of an optical or a radiation pyrometer or a thermal probe.

Pipes in steels 15M and 20M are normalised, for which purpose the welded joint is heated to 900 or 930° C, held at that temperature for 1 or 1.5 minutes for every millimetre of pipe wall thickness, and allowed to cool in quiet air. If the carbon in molybdenum steel runs not more than 0.2 per cent, normalising may be replaced by low-temperature annealing to 680 or 700°, holding at that temperature for 4 or 5 minutes for every millimetre of pipe wall thickness, slowly cooling at the rate of 50° C per minute to as low as 300° C, and then cooling in quiet air.

Welded joints in pipes from steels 12XM and 20XM are heat-treated by raising them to 930 or 950° C, holding them at that temperature for 1.5 minutes for every millimetre of pipe wall

thickness, slowly cooling down to 300° C at the rate of not over 25° C per minute, and fully cooling in quiet air.

When steel 15M is welded with a welding rod of the same analysis, the weld metal shows a higher tensile strength but a reduced impact strength in comparison with the parent metal. The mechanical properties of the deposited metal may substantially be improved by subsequent heat treatment. According to Shashkov and Vaxman, both of the Autogenous Welding Research Institute, welded joints normalised by heating with the torch to 930 or 950° C and cooling in quiet air may have a tensile strength of 45-47 kg/mm², an elongation of 13-18 per cent, and an impact strength of 10.6-14.6 kg-m/cm². Normalising in a furnace followed by high temper * may produce a tensile strength of 43 kg/mm², an elongation of 13-17 per cent and an impact strength of 16.7 kg-m/cm². The figures quoted above have been obtained at 20° C. The rate of cooling can be slowed down by suitable muffles put on the welded joint being heat-treated.

Chrome-molybdenum steels are less weldable than molybdenum types, which fact should be attributed to the chromium forming refractory oxides.

Gas-welded pipes from steel 15XMA with a diameter of 51 to 80 mm and a wall thickness of 11 to 20 mm will withstand a working pressure of up to 450 atm (gauge). The welded joints in them, when hot-peened and locally annealed with the torch, will have a tensile strength of 48-52 kg/mm² and stand a 180-degree bend test. The welding rod for them should be from steel 20XMA.

(3) Of chrome-silicon-manganese (Cromansil) steels the most common types are steels 20ХГС, 25ХГС, 30ХГС and 35ХГС, which differ between themselves mainly in carbon content. The first three grades contain 0.9-1.2 per cent Si, 0.8-1.1 per cent Mn; 0.8-1.1 per cent Cr and up to 0.4 per cent Ni. In steel 35ХГС silicon and chromium run 1.1 to 1.4 per cent each.

These steels are remarkable for high tensile strength and toughness. When properly heat-treated (e. g., quenched from 900° C in water and tempered at 640° in air), steel 20ХГС will have a tensile strength of 83 kg/mm² and an elongation of 17 per cent.

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* Heating to 900-930° C, holding at that temperature for 3 or 4 minutes for every millimetre of thickness, cooling in quiet air, re-heating to 650° at the rate of 100-150° C per hour, holding at that temperature for 3 or 4 minutes for every millimetre of thickness and cooling in quiet air.

Welded joints in Cromansil and other low-alloy steels in the pearlitic group have four principal regions differing in structure, viz.: (a) a near-weld zone with a coarse-grained structure characteristic of overheated metal; (b) an area of complete recrystallisation; (c) an area of incomplete recrystallisation and (d) a recrystallisation or temper area. The overheated metal shows a heterogenous sorbitic-troostite structures with segregated ferrite, as is typical of Widmanstätten structures. The tempering of the welded joint, while leaving the general orientation of the microstructure unchanged, results in more stable ferrite-plus-pearlite structures.

With these steels, it is possible to use butt, lap, Tee and corner joints. The thickness ratio for the parts to be butt-joined should not exceed $1 : 2$, and to be lap- and Tee-joined, $1 : 6$.

If, after heat treatment, the strength of a welded joint is of minor importance, the welding wire may be low-carbon grades like Cb-08 and Cb-08A. If after heat treatment, the tensile strength of a welded joint should be over 90 kg/mm², use should be made of Cb-18XГCA or Cb-18XMA wire.

The welding conditions for these steels should be as follows:

Metal thickness, mm	0.5	1	2	3
Rate of acetylene flow, lit/hr	50	75	150	300
Oxygen pressure, atm (gauge)	1.5	1.5	2.5	3
Welding rod diatemer, mm	1	1.5	2	2.5

The flame should be normal, and joints should be welded in a single pass in the downhand position. Vertical and overhead joints should be avoided. Prior to welding, it will be well-advised to tack the plates at every 20 or 30 mm when the metal thickness is anywhere from 0.5 to 1.5 mm, and at every 40 to 60 mm when the metal is 2 mm or more thick. Tack welds should be applied within 10 to 15 mm of the plate edge or the corner of the joint. If the welds made in Cromansil steels are to be of high quality, it is important to clean the plate thoroughly, to line up the edges accurately and to keep the gap between them constant and uniform all along the seam. Cromansil steels are susceptible to cooling cracks at the weld and in the near-weld area when allowed to cool at a high rate. This is the reason why these steels are usually welded in a manner slowing down their cooling. Upon completion of a weld, the torch should be withdrawn from it gradually and slowly, playing the preheating flame within 20 to 40 mm of the termination of the weld. To reduce warpage, both tack welds and the actual weld should be applied, working from the middle of the seam towards its ends. Welded

joints should preferably be quenched from 880° C in oil held warm at 20 to 50° C, and tempered. The hold time at 880° C should be 5 min for a thickness of 1 mm, 10 min for thicknesses from 3 to 5 mm, etc.

Whether or not a given weldment should be heat-treated depends on its designated purpose and its service conditions. The necessary information to that effect will be found in instruction charts or on the respective drawings.

3. Medium- and High-alloy Steels

Out of this class, special interest to the welder lies in chromium and chrome-nickel stainless and acid-proof steels widely employed in welded chemical equipment. The main alloying elements in them are chromium and nickel which impart the steels their special properties and structures. In turn, their structure has a direct bearing on their weldability and the properties of the welded joints made in them. Depending on their contents of chromium, carbon, nickel and other elements, these steels may be either austenitic, martensitic, semiferritic or ferritic. Those in the ferritic subdivision do not undergo phase changes in heating and cannot therefore be heat-treated. As the carbon content increases, steels become heat-treatable even though they may be high in chromium.

The austenite formed in heating may transform in cooling either to a ferrite-cementite aggregate in pearlitic or some intermediate form, or some of the austenite may be changed to martensite (as in semiferritic steels) or all of the austenite turns to martensite (martensitic steel). Austenitic steels are heat-treated only for the purpose of preventing carbide precipitation and of distributing the alloying elements more uniformly in solid solution.

When high-chromium and chrome-nickel stainless steels are heated in welding or slowly cooled through 500-700° C range, the carbon (if it runs more than 0.06 per cent) is precipitated from solid solution into the grain boundaries; chromium carbides are formed and some of the chromium is removed from solid solution to form these carbides. Hence, the grains are chromium-impoverished adjacent to the grain boundaries. Should the chromium content drop due to this cause to less than 12 per cent, the steel can be easily attacked intergranularly by corrosive reagents. The corrosion may be speeded up by the residual stresses set up at the grain boundaries due to carbide precipitation or welding. For this reason, the weld and the near-weld area are very susceptible to intergranular corrosion. As gas welding produces a broader

heat-affected zone than arc welding, the latter process should be given preference in welding stainless and acid-proof steels.

Chromium and chrome-nickel stainless steels can be made less susceptible to intergranular corrosion and grain growth by adding elements which preferentially combine with the carbon and, therefore, prevent chromium impoverishment. Titanium (about 0.8 per cent) and niobium (about 1.5 per cent) are examples of such elements. They are added in amounts increasing with the carbon content.

(1) Chromium Steels. These steels are resistant to acids and heat. Commercial grades are medium-chromium martensitic steels containing 12-14 per cent chromium; high-chromium semiferritic steels containing 13-18 per cent chromium; and high-chromium ferritic steels containing 28-30 per cent.

Chromium steels in the martensitic class are subject to air hardening and for this reason often develop quench cracks in welding. This property becomes ever more noticeable with increasing carbon content. Furthermore, these steels tend to warp in welding, as their coefficient of thermal conductivity is 0.05-0.06 cal/cm per second per degree Centigrade difference in temperature, or less than half that of plain low-carbon steels (0.13-0.19 cal/cm per second per degree Centigrade). Therefore, chromium steels should be welded with a reduced rate of acetylene flow of not over 70 lit/hr per millimetre of thickness, using a preheat, normal flame, and welding rod of the same analysis as the parent metal. Alternatively, Cв-OX18H9 or Cв-1X18H9T rods of chrome-nickel steel may be employed. This steel contains 18-20 per cent Cr and 8-10 per cent Ni, and titanium is a desirable addition. The welding operation should proceed as quickly as possible, without interruptions or reheating with the torch. Light-gauge plate should be welded by the leftward technique, and heavy gauges, by the rightward method only. The flux to be used should consist of 55 per cent boric acid, 10 per cent silicon oxide, 10 per cent ferromanganese, 10 per cent ferrochromium, 5 per cent ferrotitanium, 5 per cent titanium ore concentrate and 5 per cent fluorspar. The flux protects the chromium from burning out, helps to dissolve and remove from the weld metal the refractory oxides of chromium, titanium and niobium.

After welding, the workpiece should be given the heat treatment specified for a given type of steel. For example, articles in steel 1X13 will require quenching in air from 1,050° C and tempering in a furnace at the specified temperature. This steel has a tensile strength of 100 kg/mm², when tempered at 500° C; 75 kg/mm² at 600° C; and 60 kg/mm² at 730° C. Annealing at

780° produces a tensile strength of 50 kg/mm². Slowing down the rate of cooling in tempering produces a higher impact strength, which for this steel may be as high as 15 kg-m/cm². The workpiece should be cooled in a furnace to 600° C and in air afterwards.

Semiferritic chromium steels (such as 1X13, X18, X17K2) are susceptible to air hardening and cracking. Therefore, in welding them, it would be a sound plan to give them a preheat of 200-250° C. They are welded by the same techniques as martensitic chromium steels. The flame should be normal or slightly carburising. These steels are also subject to overheating and grain growth, and they ought to be welded at the highest possible speed. After welding, workpieces should be allowed to cool to 100-150° C and then tempered in a furnace at the respective temperature. These steels are less susceptible to cracking during the welding operation for the reason that their structure has a sufficiently plastic ferritic constituent. High-chromium ferritic steels (such as X17 and X28) show an increased grain growth in the heat-affected zone after prolonged heating. Therefore, gas welding of these steels is not desirable.

(2) Austenitic Chrome-nickel Steels (1X18H9T, X18H11Б, X18H12T, X18H12M2T, etc.). These steels stand well the attack of corrosive reagents, heat and oxidation at elevated temperatures. They are sufficiently strong and ductile and retain their initial impact strength at the very low temperatures of liquid helium (—270°). They become highly resistant to corrosion after a heat treatment which consists in heating to 1050-1100° C and quenching in water. The heating temperature increases directly with the carbon content. The complex carbides separate out of this steel in the 700-500° C range.

Chrome-nickel steels are less weldable by the oxy-gas than by the arc process. In welding them, account should be taken of their low thermal conductivity (0.04 cal/cm per second per degree Centigrade difference) and high coefficient of linear expansion (17×10^{-6}), as this results in pronounced buckling.

When gas-welded, these steels are cooled slowly through the 650-450° C range, a larger amount of chromium carbides is precipitated, and the steels lose much of their corrosion resistance.

In view of the fact that some of the alloying elements in the weld metal burn out and chromium carbides may precipitate along the grain boundaries in both the weld metal and the heat-affected zone, gas welds can be more easily attacked intergranularly by corrosive reagents than the parent metal prior to welding.

The main points to watch in welding austenitic chrome-nickel steels low in carbon are as follows: (a) the heat-affected zone should be kept as narrow as possible; (b) the rate of heating should be as high as practicable; and (c) cooling should proceed at the highest possible rate. For better control of distortions in welding, workpieces should be fitted up and welded in jigs. The welding rod to be used should be of the same composition but lower in carbon (Cв-OX18H9, Cв-IX18H9T or Cв-IX18H9Б). Alternatively, Cв-OX18H9C2 wire may be employed, containing 2 to 2.75 per cent silicon which promotes ferrite formation in the weld metal. Welding rod sizes should be as follows:

Plate thickness, mm .	under 1.5	1.5-2.5	2.5-3	3-5	6 or more
Rod diameter, mm . . .		1.5	1.5-2.5	2.5-3	3-4	5-6

To melt the rod, its tip should be dipped in the molten pool.

For flux compositions, which are mainly employed to withdraw chromium oxides into slag, see Chapter X. The flux should be dissolved in water to a paste and applied to the edges to be welded 15 or 20 minutes before the welding operation is to be started. The flux should be applied from the reverse side of the edges, as in gas welding chromium oxides may be formed at the root due to heavy heating. After welding, the weld should be thoroughly rinsed with hot water to remove surplus flux. If not removed, borate-base fluxes may give rise to an aggressive medium on the metal surface which may attack the metal intergranularly. The flame should be normal, and the rate of acetylene flow should be adjusted as follows:

Plate thickness, mm	under 1.5	1.5-2	2-4	5-7
Rate of acetylene flow, lit/hr	75	150	300	500

Long joints should preferably be welded by the step-back sequence.

After welding, the workpiece should be heated to about 1050-1100° and quenched in water. Steel plate 1 to 2 mm thick may be allowed to cool in air.

In welding austenitic chrome-nickel steels it is important that the mixture velocity and composition (one part oxygen to one part acetylene, by volume) be maintained constant. Apart from the fact that it carburises the metal and increases the heat input to the workpiece, the excess of acetylene promotes the formation of chromium carbides. With the oxidising flame, refractory chromium oxides, like Cr_2O_3, are formed. Both carburisation and oxidation are responsible for welding defects and facilitate inter–

granular corrosion. From this point of view, injector-type torches are inferior to balanced pressure torches.

Gas welding produces acceptable results in chrome-nickel steels not more than 1.5 or 2 mm thick. Heavier gauges of these steels should better be welded by the arc processes using heavily coated austenitic electrodes.

When heat-treated, gas welds in chrome-nickel austenitic steels have satisfactory mechanical properties. Still they are lower than those of the parent metal prior to welding or of argon-arc welds (Table 24).

Table 24

Mechanical Properties of the Weld Metal in Chrome-nickel Steels Welded by Various Processes

Welding process	Steel	U.T. S., kg/mm²		Elongation,%	
		parent metal	weld metal	parent metal	weld metal
Gas welding	1X18H9T	59.5	47.1	50	10.4
Arc welding, using coated consumable electrode	1X18H9T	59.5	46	53	—
Argon-arc welding	X18H11Б (Я1Н6)	59.3	73	60	38.5

When gas-weld specimens were held for 2 hours at 650° and then boiled in a mixture of sulphuric acid and vitriol, they lost their metallic ring and failed in a free-bend test due intergranular corrosion. On the other hand, argon-arc weld specimens stood a similar test without failure and showed no signs of intergranular corrosion. An explanation seems to be that the heat-affected zone in gas welding cools at a slower rate than in the argon-arc process, which fact results in the precipitation of chromium carbides into the grain boundaries.

Austenitic chrome-nickel steels with an increased carbon content (such as X18H25C2, 4X14H1B2M) are less weldable than chrome-nickel steels low in carbon. Among other things, their carbides are more apt to precipitate along the grain boundaries, giving rise to hot cracking in welding. To avoid cracking, these steels require a preheat of 280-300°C and the use of CB-0X18H9 welding wire. These steels are better welded by the arc than the oxy-gas process. After welding, the workpiece and welded joints in it should be heat-treated. In the case of steel X18H25C2, this in-

volves heating to 1150-1100° C and cooling in air, and for steel 4X14H14B2M heating to 820-850° and slow cooling.

(3) Austenitic Manganese Steels. This class embraces steels containing 11-14 per cent Mn and 1-1.3 per cent C (Hadfield steel). When heated to 1050° C and quenched in water, the steel acquires high mechanical properties (U. T. S. = 100-120 kg/mm² and 40 per cent elongation). When cooled slowly, the steel loses much of its strength due to the precipitation of manganese and iron carbides along the grain boundaries. The steel is extremely wear and shock resistant and goes to make rails, frogs, crossings, the wearing parts of steam shovels, etc.

The main difficulty encountered in welding this steel is reduction in the ductility and strength of the parent metal, especially where the metal is heated to above 1200° C, as the coarse-grained structure is formed and cracks are likely to develop there. For these reasons, the gas process is only used for the repair and reclamation of worn parts from manganese steel, and not for production welding.

The parts to be welded are given the single-V preparation with an included angle of 90° C. The double-V (or X) preparation is not used, as it involves the reheating of the workpiece during the welding operation — an undesirable corollary for this type of steel. The rate of acetylene flow is usually set at 130-150 lit/hr per millimetre of thickness. There should be a slight (5-10 per cent) excess of acetylene in the flame, so that the manganese will not be heavily oxidised during welding. The welding rod to be used is of the same analysis as the parent metal. Use may also be made of Cʙ-0X14 and Cʙ-0X13 wire, chrome-nickel Cʙ-0X18H9 wire and chrome-nickel-manganese Cʙ-X20H10Γ6 and Cʙ-X20H10Γ6A wire.

Welds should be deposited in a single pass in the downhand position. As large quantities of gas are evolved in the molten pool, weave beads should be employed and deposited in short increments. For better results, welds should be peened while hot. Still better results can be obtained by heating the weld metal to 1050-1100° C and quenching in water.

CHAPTER XIII

CAST IRON WELDING

1. Properties of Cast Iron

Cast iron is an alloy of iron and carbon. Theoretically, carbon in cast iron may run anywhere from 1.7 to 6.7 per cent. Commercial grades of ordinary cast iron, however, contain 2.6 to 3.7 per cent carbon. Cast iron is almost universally used for forms that must be shaped by casting, especially where considerable weight is desirable or is not objectionable.

There may be grey cast iron, white cast iron, malleable cast iron, inoculated cast iron, alloy-type cast iron — all differing between themselves by the presence and relation of the microconstituents — ferrite, pearlite, graphite, cementite and phosphide eutectic in the metal.

Grey cast iron is the most commonly used of them in engineering. The graphite grains in it are larger, and the fracture is greyish in colour. The carbon is present as graphite, and only a small proportion of it is combined carbon, or cementite (Fe_3C). The fact that the free carbon is distributed in the ferrite makes grey iron easily machineable. On an average, the analysis of grey iron is 3-3.6 per cent C, 1.6-2.5 per cent Si, 0.5-1 per cent Mn, up to 0.12 per cent S and up to 0.8 per cent P.

Grey Iron Properties
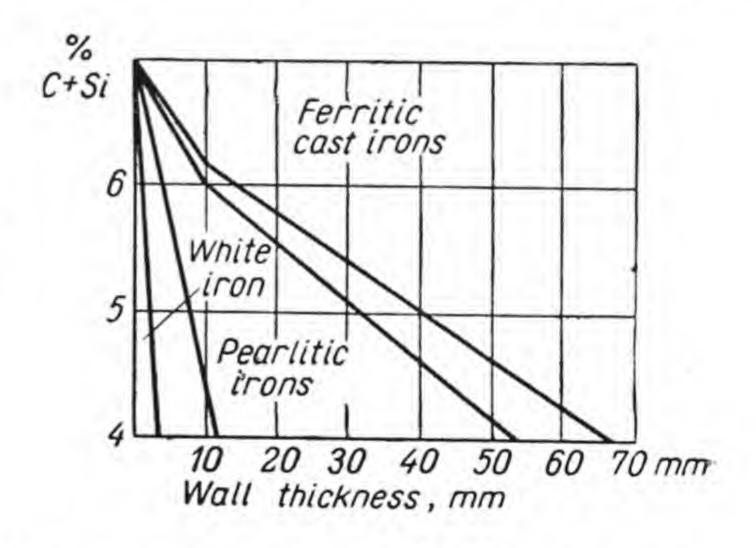
Density, g/cm^2	6.9-7.5
Thermal conductivity at 20° C, cal/cm per sec per degree Centigrade difference	0.06-0.08
Coefficient of linear expansion between 20 and 600° C	10×10^{-6}
Melting point, degrees Centigrade . .	1100-1250
Linear contraction, per cent	0.7-0.8

The segregation of carbon as graphite in the liquid phase is governed by the silicon content, and in the solid phase by the carbon content. According to K. V. Lyubavsky, complete graphitisation in grey iron containing 3-3.2 per cent carbon under weld-

ing conditions takes place when the silicon in the deposited metal runs over 5 per cent. Silicon reduces the solubility of carbon in both solid and liquid solutions, as it shifts the eutectic and eutectoid points up and leftward — towards the more elevated temperatures and lower carbon concentrations. Carbon promotes graphitisation by increasing the number of crystallising nuclei and raising the temperature of the eutectic transformations of the alloy.

Aluminium in cast iron acts in a similar way. High rate of cooling causes the carbon to separate out as cementite, thereby making the fracture mottled or white. When the rate of cooling is low and graphitisers are present in sufficient quantities, the cementite decomposes, graphite carbon is rejected and grey iron is produced. Fig. 105 relates the rate of cooling and the cast-iron

Fig. 105. The effect of C and Si contents and the rate of cooling (wall thickness) on the structure of cast iron

structures. It will be noted that reduction in wall thickness (i. e., an increase in the cooling rate) results in white iron.

White iron is harder and more brittle than grey iron. It owes its hardness to the presence of cementite and pearlite. In engineering, white iron is only used as the source material for the production of malleable iron.

In a welded joint, the desired degree of graphitisation and, consequently, the desired iron structure can be obtained by controlling the carbon and silicon content in the deposited metal and maintaining the appropriate rate of cooling.

Next to grey iron in their importance to engineering are a variety of inoculated and alloy grades of cast iron: high-strength, antifriction, corrosion-resistant, heat-resistant, high-silicon, silicon-molybdenum, high-chromium, etc.

Alloy-type high-strength iron is produced by adding some alloying elements while grey cast iron is poured into moulds. These additions are silico-calcium, ferro-silicon, silico-aluminium,

magnesium and magnesium alloys, etc. As a result, the graphite in straight inoculated iron segregates in the form of fine flakes, while in high-strength iron it shows as globules. These irons are comparable in tensile strength with steel and possess sufficient ductility (a tensile strength of 50-55 kg/mm², an elongation of 1.5-3 per cent and an impact strength of 1.5-3 kg-m/cm²).

The automotive industry widely uses malleable iron produced by subjecting ordinary white iron castings of the desired shape to a special heat treatment (malleablising) in an oxidising or neutral atmosphere at 800-1000° C.

Both alloy-type and malleable irons are less weldable than grey iron, as they lose their original properties in the heat-affected zone. Better results are yielded by brazing them with brass.

Sulphur is a harmful impurity in cast iron as it reduces its weldability and strength, makes the iron less fluid when molten, and increases its shrinkage. Sulphur also retards graphitisation. Phosphorus moderates the harmful effect of sulphur, increases the fluidity of molten iron, improves its weldability, but, at the same time, makes it hard and brittle. This is the reason why sulphur and phosphorus in cast iron should be kept within the specified limits.

2. Cast Iron Welding

For all the many arc-welding processes developed, especially for grey iron, gas welding still remains the basic method, as it produces high-quality full-strength welds.

Cast iron may be welded when a casting or a ready part has to be repaired, cracks or blowholes filled up, a broken part reconditioned, etc.

Of all the iron, grey foundry iron is gas-welded best. The prolonged and uniform heating characteristic of gas welding promotes graphitisation in the weld metal.

The principal difficulties encountered in cast iron welding follow.

(1) The segregated graphite may at some places change to cementite so that the weld metal there acquires the structure of white iron and would not be machined. This is known as the chilling effect and may take place in both the weld and parent metal, raised close to melting point. The chilling effect is attended by the burning out of carbon and silicon. Then rapid cooling coupled with the deficiency of carbon and silicon results in a white hard metal.

(2) When it cools, a cast-iron part, especially one of odd shape, is subjected to considerable thermal strains and stresses due to

the thermal gradient and contraction, and cracks may develop in both the weld and parent metal.

(3) Cast iron has a relatively low melting point and quickly passes from the solid to the liquid state and back. Due to this, the gases dissolved in the metal have no time to escape from the puddle, and the weld sometimes has porosity. The fact that molten cast iron is very fluid necessitates its welding solely in the downhand position.

(4) The refractory film of silicon and manganese oxides formed in welding has a higher melting point than the parent metal and prevents the metal grains from complete fusion.

Cast iron welding may use preheating which may be general or local.

The edges are usually given the single-V preparation with an included angle of 90°. The double-V (or X) preparation is used more seldom. In welding up cracks in an iron casting, the defective area may be either melted out by the torch or removed by oxy-flux gouging. The welding rods to be used should be 6, 8, 10 or 12 mm in diameter and 400 to 700 mm long. Their analyses are summarised in Chapter X.

For substantial parts welded with a preheat, Grade A welding rods should be used, containing 3-3.5 per cent silicon and 0.2-0.5 per cent phosphorus. In other cases, Grade B rods will do.

The refractory oxides of silicon and manganese as well as of iron formed in welding are slagged through the use of a suitable flux.

In welding, the tip of the welding rod should be dipped into flux from time to time; in addition, some flux is added to the molten pool. The welding flame should be normal or have a slight excess of acetylene. The oxidising flame would increase the oxidation of the silicon and manganese. Welding may only be carried out in the downhand position. Prior to actual welding, the edges should be thoroughly and uniformly preheated as a precaution against porosity, incomplete penetration and lack of fusion. Substantial iron parts should be welded with two torches, one torch being used for preheating. The heat input is adjusted so that the rate of acetylene flow is 100 to 120 lit/hr per millimetre of plate thickness (per torch). The weld pool should be continually puddled with the rod end so as to facilitate the escape of the gases from the molten metal, or much porosity will be left in the weld.

Odd-shaped parts should be given overall preheat as a precaution against internal stresses and cracking due to the temperature gradient. In such cases, parts should be given a preheat

of 500-600° C in a suitable furnace, hearth or pit. Small parts may be preheated with the torch.

Where the section to be welded is constrained in one direction only, and the constraining elements can be easily identified, it will be sufficient to preheat only the constraining elements. They are heated just to cause them to expand by the same amount as the area being welded (Fig. 106a). This usually involves a preheat of 500-600° C.

Where the area to be welded is bounded by the parent metal on all sides, the part has to be given an overall preheat
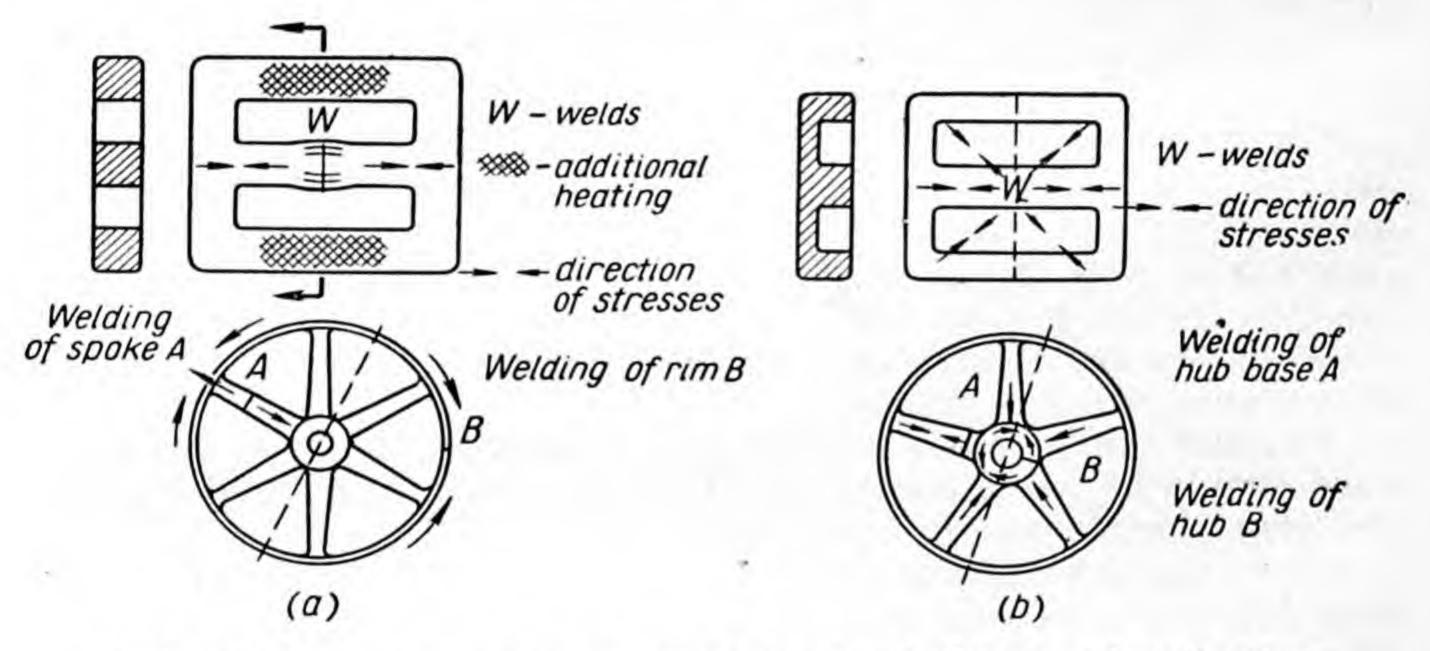
Fig. 106. Cast-iron welding with (a) local and (b) general preheating

(Fig. 106b). Local preheating may be accomplished with torches, blow-lamps, charcoal or coke. In the latter case, the area to be preheated is enclosed by a sheet-iron box with holes to admit air; the box is filled with charcoal or coke, and the fuel is fired. In local preheating, it is essential to see that all the parts of the workpiece are heated and cooled uniformly and, what is most important, gradually. For this reason, the areas of the workpiece lighter in section should be heated in cooling so as to prevent them from cooling faster than do the areas heavier in section.

In large-scale production welding, say, in the fabrication of cylinder blocks for automobile engines, use is made of multi-chamber or conveyor-type furnaces (Fig. 107) in which the parts to be welded are heated and fed continuously to the scene of welding operations. It is important that a heated part be allowed, upon welding, to cool slowly with the furnace. This produces the uniform

structure of grey iron and prevents cracking. Instead of cooling in the furnace, quantity parts may be held in heat-insulated chambers. In that case, the rate of cooling is slowed down due to the heat stored up in the part itself.

Cast iron may advantageously be welded with natural gas which, when burned with oxygen, produces a flame less hot than the oxy-acetylene flame. As a result, the metal is heated gradually and uniformly, with less silicon burned out in welding — a condition vital for the soft grey iron structure to be obtained in the
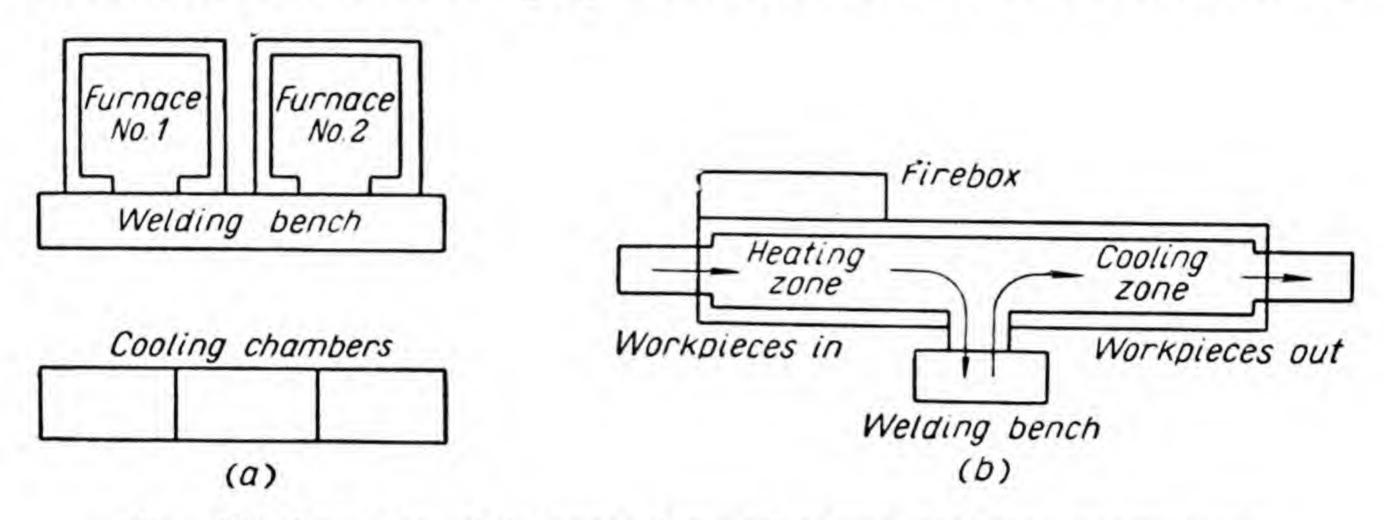
Fig. 107. Location of furnaces for preheating cast-iron workpieces:
a — multichamber furnace; *b* — conveyor type furnace

deposited metal. Natural gas as fuel for welding is usually supplemented by cast-iron welding rods whose analysis includes 3-4 per cent silicon or 0.1 per cent aluminium to promote graphitisation. The flux to use may be roasted borax or a one-to-one mixture of roasted borax and charcoal, or 50 per cent borax, 47 per cent sodium bicarbonate and 3 per cent silica.

Recently, a technique has been developed for cast iron welding, which involves the use of a special flux paste. The place to be welded is thoroughly cleaned, given a local or an overall preheat of 300-400° C with a reducing torch flame, painted with the paste along the edges to be welded, and the edges are raised to 750-790° C with a normal or a reducing flame. The paste is melted to produce a blanket of vitreous mass over the weld. The cast-iron welding rod is also dipped in the paste. The temperature of the weld should not exceed 800° C, and filler metal is deposited in passes of small thickness. After the weld is completed, another coating of paste is applied to it, while the workpiece is allowed to cool in sand, lime or asbestos. Due to the slow and relatively low heating of the part, steady cooling and the fluxing action of the paste, graphitisation in the weld metal proceeds under favour-

able conditions, and the deposited metal has the fine-grained structure of grey iron. The analysis of the welding rod usually is 3-3.8 per cent carbon, 2.8-4.1 per cent silicon, 0.52-0.8 per cent manganese, 0.4-0.6 per cent nickel, 0.1 per cent chromium, and 0.15-0.2 per cent titanium. The composition of the paste is 8 per cent TiO_2, 10 per cent KNO_3, 8 per cent CaF_2, 45 per cent $Na_2B_4O_7$, 3 per cent FeTi, 21 per cent NaF and 5 per cent LiCl (dissolved in ethyl alcohol in a 5 : 12 proportion). The powdered mixture is screened through a 100-mesh sieve and mixed with the bright stock grade of cylinder oil in a proportion of 2 parts of oil to 3 parts of the mixture. Then the alcohol solution of LiCl is added to the resultant mixture. The paste should be kept in a hermetically sealed can for not more than 25 days.

3. Cast Iron Brazing

Cast iron is best brazed with brass welding rods. For one thing, this calls for low heating, and the original structure of the metal can be easily retained. For another, brass rods produce welds which are superior in ductility and plasticity to the parent metal. This is why the brazing technique is widely used in the repair of critical parts from cast iron. The technique may be especially recommended for alloy-type globular high-strength and malleable irons, as the structure and properties of the original metal remain unaffected.

Cast iron brazing usually involves either overall or local pre-heating. Welding rods usually have a melting point of 850-900° C as compared with cast iron's 1200° C. Therefore, the edges to be welded are only heated to 900-930° C and not melted, and the molten metal of the rod is deposited in the joint. The weld thus produced is ductile and stands up well to both tension and impact loads. Its ultimate tensile strength is 35-40 kg/mm² and its elongation, 8-15 per cent. No dangerous stresses are developed in the part, as the heat applied is not intense.
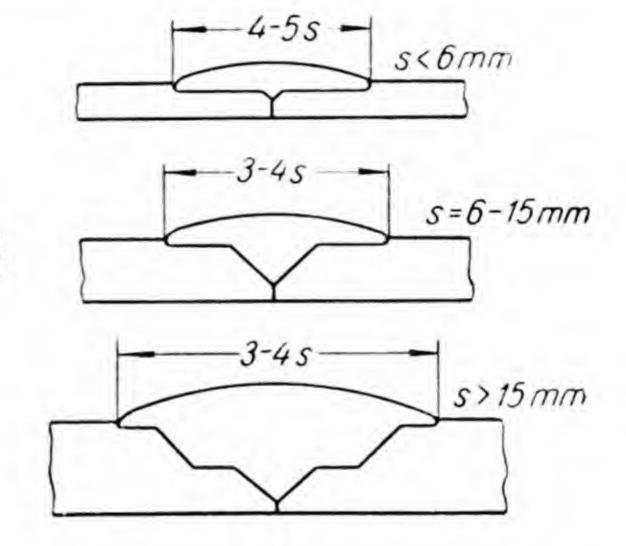
Unfortunately, the fusion area in this type of weld is mechanically inferior to the parent metal and the weld deposit. To strike an even balance between the two, the fusion area has to be made larger in cross-section and the welds shown in Fig. 108 should be employed. In cast iron brazing the rate of acetylene flow is about 75 lit/hr per millimetre of thickness. The flux to use is 70 per cent melted borax, 20 per cent common salt and 10 per cent boric acid. As an alternative, borax alone or a one-to-one mixture of borax and boric acid may be used. The brazing rod may have the following analysis: 58-59 per cent copper, 38-40 per cent zinc, 0.9-1.1 per cent tin, 0.5-1 per cent iron and 0.4-0.8

per cent manganese. Sometimes the brass may contain up to 0.5 per cent nickel and, as deoxidizers, up to 0.12 per cent phosphorus and up to 0.25 per cent silicon. This purpose may also be served by brass ЛО-62-1.

Prior to welding, the edges should be bevelled to an angle of 45° and roughed with a chisel to produce a surface to which brass will adhere. The edges are preheated with an oxidising flame to burn the graphite coating away from the fusion faces. Then a short length of seam is heated to 850-900° C (the melting point of brass), and the heated surface is "tinned" with a layer

Fig. 108. Joint preparation and cross-section in the case of brass brazing of cast iron

of brass. To do this, the fusion edges should be sprayed with flux and rubbed with the brazing rod until it melts. After tinning, the balance of the vee is built up with brass. The flame should be slightly oxidising (to prevent zinc burn-out and porosity in the weld), with one part acetylene to 1.3-1.4 parts of oxygen. The less hot part of the flame should be used, by holding the inner cone farther away from the weld pool than usually. It is important not to overheat the metal at the weld.

4. Alloy and Malleable Cast Iron Welding

Cast irons containing nickel, chromium, copper and other alloying additions are welded in a usual manner, using welding rods of as much the same analysis as the parent metal as possible. If they are not available, use may be made of the welding rods employed for grey iron. The weld metal will then differ, however, from the parent metal in analysis to a marked degree. Indeed, it may lose some of its special properties (such as corrosion resistance).

Inoculated cast iron may be welded with rods containing up to 0.1 per cent magnesium. Cracking can be prevented by giving the workpiece a preheat of 500° C.

Malleable iron is best welded with rods which produce the white-iron structure in the metal deposit. Upon welding, the workpiece is heat-treated in an annealing furnace to make the weld metal malleable.

Parts from oxidised iron are repaired in short increments, each increment being thoroughly cleaned of the oxide layer until the sound metal shows up. This is done by heating the area to be welded with the torch, which is then sprayed liberally with flux, the oxides are skimmed with the welding rod, and filler metal is deposited on the clean surface. This procedure is repeated on all of the increments of the seam which is then built up with filler metal across the entire thickness of the workpiece. It will be sound practice to repair oxidised iron by brass brazing.

WELDING OF NONFERROUS METALS

The welding of nonferrous metals runs into a number of difficulties which are due to some of the properties of the metals. These, among other things, are the tendency to form a refractory oxide film, gas absorption by molten metal vaporisation on the metal with the resultant porosity, high thermal conductivity and heat capacity, great linear expansion in heating, an appreciable shrinkage during the transition from the liquid to the solid state, the tendency of the grains to grow in heating, etc.

The main difficulty is the intensive oxidation of nonferrous metals in the course of welding. The oxides thus formed prevent close adhesion between the grains in the weld metal, thereby reducing its strength and ductility. To avoid this, it is vital to employ efficient fluxes which will deoxidise the puddle and dissolve the oxides to produce easily melting slags protecting the molten metal from further oxidation.

Gas welding is a universal process equally applicable to all nonferrous metals of importance to engineering, but it is expensive and slow. In some cases, especially on heavy sections, the weld metal shows impaired mechanical properties due to overheating, oxide inclusion, weld porosity and some other thermal and metallurgical factors inherent in gas welding. This has spurred the search for other processes more efficient as far as nonferrous metals are concerned. These are, above all, submerged-arc and inert-gas shielded arc welding which lends itself readily to mechanisation and automation. Still, gas welding remains to be widely employed in industry, mainly for copper-base alloys (especially brasses).

From what has been stated above it is not hard to see that the selection of a proper welding procedure and technique is of primary importance in the welding of nonferrous metals.

1. Copper

The properties of copper vary with its purity, machining and heat treatment.

Bismuth, lead, sulphur and oxygen are insoluble in copper. The first two form eutectics of low melting point which are segregated along the grain boundaries and make the copper red-short and brittle. Sulphur and oxygen also form low-melting eutectics embrittling the copper. When the sulphur (as Cu_2S) runs more than 0.1 per cent, the copper also becomes red-short. Oxygen is usually present in copper as a constituent of Cu_2O produced by the reaction $4Cu + O_2 = 2Cu_2O$. The $Cu_2O - Cu$ eutectic will be formed when the copper contains 0.38 per cent O_2 and 3.4 per cent Cu_2O. The latter is particularly ready to form at temperatures close to the melting point of copper.

In welding copper M3, the most commonly used grade in the Soviet Union, the cuprous oxide (Cu_2O) goes into the Cu_2O-Cu eutectic which segregates along the grain boundaries in the intermediate zone, thereby reducing the mechanical strength of the welded joint. In microphotographs the cuprous oxide shows itself as numerous inclusions. As the welding speed is increased, the area where the Cu_2O-Cu eutectic segregates becomes narrower. The addition of deoxidisers to the welding wire and flux also diminishes the width of the area where the eutectic is likely to segregate. Molten copper containing cuprous oxide absorbs hydrogen and carbon monoxide which react with the cuprous oxide,

$$Cu_2O + H_2 = 2Cu + H_2O;$$

$$Cu_2O + CO = 2Cu + CO_2.$$

The water vapour thus formed expands to produce fine cracks in the metal deposit. This is known as the hydrogen corrosion of copper.

The amount of cold work and the temperature of heating also affect the mechanical properties of copper. As the amount of cold work increases, the ultimate tensile strength of copper increases, but its ductility (percentage elongation) is reduced. Temperature reduces the ultimate tensile strength of copper. For soft copper it is 22-23 kg/mm², and for cold-work hardened copper, 37-42 kg/mm². The other physical properties of copper are as follows:

Melting point	1080-1083° C
Density	8.9 g/cm³
Linear expansion at 20° C . .	17.7×10^{-6}
Thermal conductivity	0.95 cal/cm per sec per deg. C diff.
Linear shrinkage of molten copper in solidification . . .	2.1 per cent

As copper possesses high thermal conductivity, more heat has to be imparted to the welding area than in welding, say, steel. Excessive heat input, however, may overheat the weld metal and adjoining areas in the parent metal with the resultant grain coarsening and reduced strength. For copper in thicknesses under 10 mm, heat input should be chosen according to Eq. (XI. 3) and correspond to a rate of acetylene flow of 150 lit/hr per millimetre of thickness.

On heavy gauges, the rate of acetylene flow has to be raised to 200 lit/hr per millimetre of thickness. On thicknesses over 10 mm, it will be good practice to employ two torches: one for preheating and the other for welding. Alternatively, the workpiece may be welded from both sides. In such cases, the flame output of the preheating torch should be equivalent to a rate of acetylene flow of 150-200 lit/hr, and for the welding torch, 100 lit/hr per millimetre of plate thickness. The plates to be welded should be placed on sheets of asbestos to reduce heat abstraction.

In the case of heavy sections, it will be a good plan to cover them with asbestos sheets also from above, and on either side of the joint. The normal flame may only be used. An oxidising flame is extremely harmful because of the intensive oxidation of the copper. An excess of acetylene promotes the formation of pores and cracks due to the fact that the cuprous oxide (Cu_2O) is reduced by H_2 and CO appearing in excess in such a flame. The inner cone should be 3 to 6 mm away from the surface of the puddle, without touching the molten metal. For better heat penetration, the torch should be held at a nearly right angle to the surface of the plate.

The flame envelope should never be directed away from the molten metal, or else the copper will be strongly oxidised at that place.

The welding operation should be carried out with as few interruptions as practical, and with the highest possible speed, for this appreciably reduces the amount of cuprous oxide forming due to the welding heat. This is the reason why it will be well-advised to use concurrent heating, as it speeds up the welding operation. On the other hand, tack welds should not be employed, as they may give rise to cracking during actual welding where the tack welds have been applied. Instead, the parts to be welded should be fitted up in jigs and fixtures.

In welding, the edges and welding rod should be melted simultaneously, without leaving the melted edges uncovered by filler metal even for a short time. When the rod is melted while the edges are still in a pasty state, the filler metal will easily fill the groove. Root runs should be avoided because of cracking.

The plates to be welded should be fitted up so that the gap at the end of the joint is 2.5 to 4 per cent of the joint length. The weld should be started one-third of its length from the joint end. The longer increment should be welded first, and the shorter increment afterwards. Plates less than 3 mm thick, may be square-butt welded. Heavier gauges require bevelling to 45° and a root face equal to one-fifth of plate thickness. Lap joints should be avoided, as the resultant welds usually lack in fusion due to the high thermal conductivity of copper.

The filler metal to use is wire from pure (electrolytic) copper (Grade M1 or M2), or copper with additions of phosphorus or silicon.

Rods from electrolytic copper are used to weld workpieces 1 or 2 mm thick. For sections 3 to 10 mm thick, it is recommended to employ copper rods with an addition of 0.2 per cent phosphorus, while on thicknesses over 10 mm, the filler metal should be copper wire containing 0.2 per cent phosphorus and 0.15-0.30 per cent silicon. Phosphorus increases the fluid of the molten weld metal. In addition, it deoxidises copper. The cuprous oxide is reduced by the phosphorus according to the reaction:

$$5Cu_2O + 2P = P_2O_5 + 10Cu.$$

The welding rod should melt quietly, without sputtering. When drops of molten copper break away from the rod, its tip should remain bright and rounded.

Silicon is a deoxidiser which reduces the fluidity of molten metal, and the volume of the puddle, and enables copper to be welded in the vertical position. It is advantageous to use welding rods with a melting point 100° C below that of the parent metal. The excess of the deoxidisers (P and Si) in the rod results in the porosity and reduced mechanical strength of the weld metal.

It is important to select the right size of welding rod. With too small a diameter, the filler metal may be burned, while with oversize diameters there may be lack of penetration in the parent metal.

The following sizes of welding rods may be recommended:

Metal thickness, mm	under 1.5	1.5-2.5	2.5-4	4-8	8-15	over 15
Rod diameter, mm	1.5	2	3	5	6	8

Copper should be welded with a flux in order that the metal can be deoxidised properly and the oxides formed in the process removed into slag. If the welding rod used contains no phosphorus, some phosphorus (as sodium phosphate) should be included in the flux. The flux should be applied so as to cover the weld

pool, parent metal within 5 or 6 mm of the edges and the reverse side of the weld. Good results have been obtained with a pasty flux consisting of 50 per cent borax, 20 per cent charcoal, 15 per cent SiO_2, and 15 per cent Na_2HPO_4. The ingredients are mixed with soluble glass and the paste is applied to the welding rod already pickled with nitric acid and rinsed with water. The rod should be preheated to 50 or 60°, and the flux should be applied in two layers, totalling between themselves 0.5 to 1 mm.

After it has been completed, the weld and the near-weld area should be peened to make the seam dense and to refine the grain. On thicknesses under 5 mm, this may be done in a cold state. On greater thicknesses, the metal should be hot (200-300° C). When properly applied and peened, the metal deposit may have a strength of 17-22 kg/mm². The reinforcement is made higher than is usual, in order that the bulk of porosity may be brought out of the weld metal. Heavy peening should be avoided, as the cold work may lead to cracking due to the local losses in plasticity. After peening, the weld should be annealed with the torch to 500 or 550° C and quenched in water. No peening should be resorted to at higher than 500° C, as copper at that temperature loses much of its strength and may develop cracks. The annealing and quenching after welding improve the plasticity of copper, as this treatment refines the structure of the weld metal.

2. Brasses

Brasses are alloys of copper and zinc. The zinc usually runs from 20 to 55 per cent. The brasses containing under 10 per cent zinc are called tombacs, and those carrying 10-20 per cent zinc, semitombacs. The zinc content has a decisive effect on the structure and workability of a brass.

Brass containing under 37 per cent zinc may be considered a single-phase system, mainly the alpha-phase which is a substitutional alloy, or solid solution, of zinc in copper. For this reason, it has the same crystal structure as copper.

All-alpha brasses lend themselves readily to pressing and stamping when cold. When sufficiently pure (especially with respect to lead) they can also be conveniently pressed or stamped when hot. A heating of over 500° C markedly reduces the strength of alpha-brasses, and so the welds in it should preferably be peened in the cold state.

With zinc running from 37 to 45 per cent, a second constituent appears in the brass known as the beta-phase, or beta-solid solution. The beta-phase is body-centred cubic, is harder and less ductile when cold, but is sufficiently ductile when hot.

In practice, due to insufficiently slow cooling after welding, beta-solid solution may be retained in the weld metal with zinc running as low as 32.5 per cent.
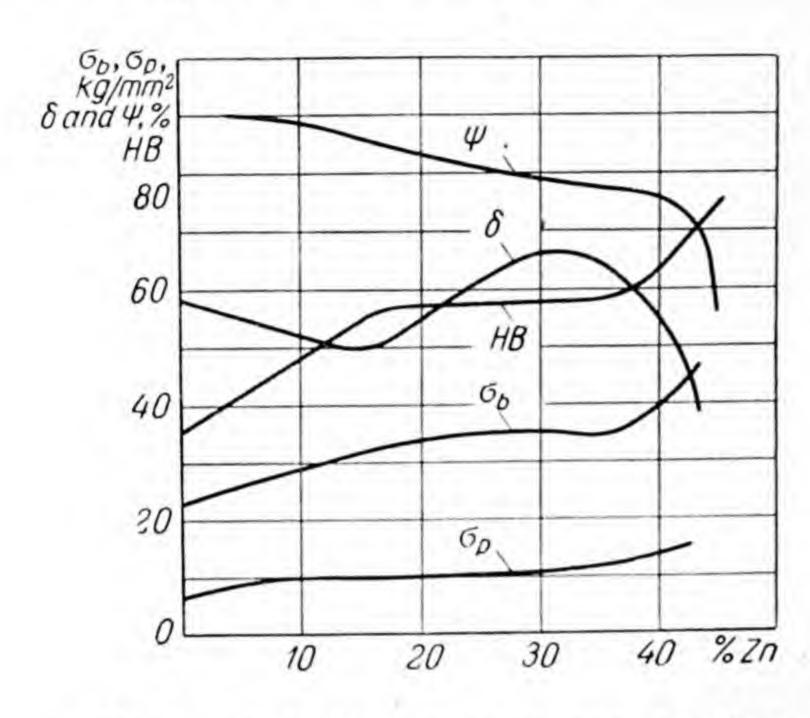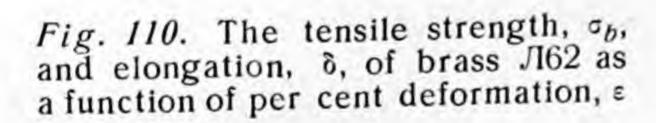
Fig. 109. The effect of zinc content on the mechanical properties of copper-zinc alloys (after 40 per cent reduction and annealing at 550° during 2 hrs):
σ_b — ultimate tensile strength, kg/mm²;
σ_p — proportional limit, kg/mm²;
δ — percentage elongation;
Ψ — reduction of area, per cent;
HB — Brinell hardness number
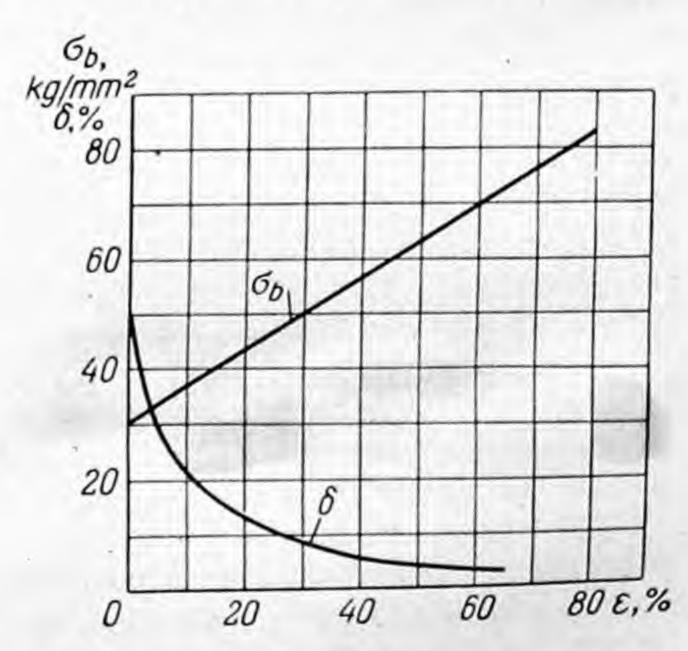
Alpha + beta brasses are of the cast type. They cannot be machined when cold, owing to lack of plasticity. If a two-phase

Fig. 110. The tensile strength, σ_b, and elongation, δ, of brass Л62 as a function of per cent deformation, ε

brass is to be hot-worked successfully, the alloy should be heated until it consists entirely of beta solution. The range of red short-ness for alpha + beta brasses lies between 350° and 750° C. The mechanical properties of brasses are governed by their zinc content (Fig. 109) and the amount of cold work they have received (Fig. 110).

The specific gravity of brasses is 8.85 to 8.5; the coefficient of linear expansion is 17×10^{-6} to 20×10^{-6}, thermal conductivity averages at 0.3 cal/cm per second per deg. C difference.

Straight brasses (binary copper-zinc alloys), containing over 6 per cent zinc tend to crack spontaneously (called season cracking) after cold work due to locked-up stresses, especially when the surrounding atmosphere contains moisture and vapours of ammonia or mercury salts. The susceptibility to season cracking increases directly with the zinc content. As a precaution, articles and welds in brasses should be stress-relieved for 1 hour at 275-300° C.

Brasses show the highest ductility with zinc running 30-32 per cent, when the β-phase is practically absent. As the zinc content increases to 50 per cent, the quantity of the β-phase increases, and the ductility of the metal is reduced.

Brasses which can be reduced to 20 per cent of their original volume are classed as semihard brasses. When the reduction in volume is 40 to 50 per cent, they are classed as hard.

After annealing it is usual to pickle brasses with a 10-per cent solution of sulphuric acid at 20 to 50° C to remove scale. The pickling is then followed by rinsing with water and drying.

When α-brasses are annealed at higher than 650° C, they may develop a coarse-grained structure, which is undesirable, as in the subsequent bending or stamping, the surface will have the appearance of fish-scale.

The most widely used brasses in the fabrication of chemical apparatus are Л90 (tombac), Л80 (semitombac), Л68, Л62 and ЛМЦ58-2 (classed as manganese bronze), and ЛЖМЦ69-1-1 (classed as iron-manganese bronze).

The melting point of brasses progressively goes down as the zinc content increases. For Л90 it is 900-950° C, and for Л62, 880° C.

The main difficulty in brass welding is the vaporisation of zinc from the puddle. Therefore, in selecting a welding procedure, the principal criterion should be whether it can reduce the loss of zinc.

The loss of zinc is the least with gas welding. This is due to the fact that the gas welding process makes it possible to control heat distribution in the welding zone more accurately than is possible with other processes.

The vaporisation of zinc in brass welding is due to the fact that zinc starts boiling at 905-906° C, i. e., much below the melting point of the alloy. No copper vaporises from the alloy, for the pressure of copper vapours at the welding temperature is close to zero.

The pressure of zinc vapours increases with the temperature of the alloy, and zinc vaporisation increases. Simultaneously with zinc vaporisation, its vapours are oxidised to zinc oxide according to the reaction

$$2Zn + O_2 = 2ZnO.$$

The porosity often formed in the metal deposit in brass welding is to an appreciable degree due to the fact that copper-zinc alloys have a short temperature range between the liquidus and the solidus (the solidifying range).

According to G. A. Asinovskaya of the Autogenous Welding Research Institute, the principal cause of porosity in brass welds is the hydrogen of the welding flame. Present in an active atomic state, the hydrogen dissolves in the molten metal; as the molten metal solidifies, the hydrogen escapes from the metal to form molecules which appear as gas bubbles (gas pockets) in the metal. These bubbles serve as reservoirs for zinc vapours; the pressure in the bubbles gradually rises, the bubbles expand and finally pores are left in the deposited metal.

Some pores may also be formed by other gases evolved in the metal by the reduction of the oxides. Therefore, if the metal deposit is to be free from porosity, the parent metal must be deoxidised in welding as much as practical.

Zinc oxide is the principal oxide formed in brass welding. It does not dissolve in molten metal and boils at about 1900° C. In welding, it is usually reduced by silicon which has a greater affinity for oxygen than zinc.

From the puddle, the zinc oxide can be slagged by a flux. The fluxes used in brass welding are boron-base compounds (borax, or boric acid). Their compositions are presented in Chapter X.

It should be noted that the presence of silicon in the puddle sharply reduces zinc vaporisation by producing a protective film, mainly of silicon oxides, on the surface of the weld pool. The film has a great surface tension and prevents the zinc vapours from escaping into the atmosphere.

The flame to be used in brass welding should be strongly oxidising (with an oxygen-to-acetylene ratio of 1.3 or 1.4), so as to suppress the reactions leading to the evolution of free hydrogen in the flame.

If a brass article has a surface film of oxides, the cleaning of the edges with a brush or emery paper should be supplemented by pickling in a 10-per cent solution of sulphuric acid, rinsing with hot water and wiping to dryness with a cloth.

Heat input should be calculated by Eq. (XI. 1). Both excessive and insufficient heat input should be avoided, as both, though for

different reasons, increase zinc vaporisation. The chances that the weld may be porous are higher with heavier gauges. To speed up the welding of the heavier sections, it will be sound practice to heat the edges concurrently, using another torch.

Zinc vaporisation may be reduced by holding the inner cone of the flame 7 to 10 mm away from the metal surface, the flame envelope being directed at the welding rod rather than at the parent metal. The tip of the rod should be dipped into flux from time to time, and some flux should be sprinkled on the weld pool and the edges. The welding speed should be maintained as high as practical.

The welding rod may be of the same analysis as the parent metal, but better results have been obtained with filler metal containing up to 0.5 per cent silicon which acts as deoxidiser for the metal deposit. For the brasses of the Л62 class it pays to use ЛК62-0.5 welding wire containing 62 per cent Cu, 0.45-0.5 per cent Si, and the remainder zinc. With this filler metal, the puddle is efficiently cleansed, the metal deposit is dense and the zinc burn-out is reduced to a minimum. This grade of filler metal has been developed by the Autogenous Welding Research Institute. When tin is used as the deoxidiser in the welding rod, its addition in excess of 0.7 per cent usually reduces the ductility of the weld metal.

Brasses should preferably be welded from one side only to avoid rewelding and sealing runs on the reverse side, as reheating adds to the losses of zinc through burn-out, promotes grain growth and may even result in cracks in the weld and the heat-affected zone. For the same reason, lap joints should be avoided. The edges of brass workpieces may be bent only when their temperature is outside the red shortness range (400-700° C), or else cracking may result. Cracks may also be produced by peening at temperatures close to melting point, i. e., 900° C. Peening is usually resorted to in order to make the weld metal dense and to enhance its mechanical strength. Sometimes, the reinforcement is peened down until it is flush with the parent metal. If the brass contains less than 60 per cent copper, the weld may be peened while hot (at over 700° C). Where copper runs over 60 per cent, peening should be carried out when cold. For the higher ductility of the metal deposit, the peening operation may be supplemented by an anneal of 600-650° C and slow cooling. This type of treatment produces a fine-grained structure in the weld and eliminates the effect of cold-work hardening.

Gas welds made in brass Л62 with a thickness of 3 or 4 mm, using Л62 welding wire, have an ultimate tensile strength of 30 or 33 kg/mm², while on plate 10 mm thick the figure is 23 or

24 kg/mm². Gas welds made in Л62 brass plate 4 mm thick weld-
ed with ЛК62-0.5 wire may have an ultimate tensile strenght of
38 kg/mm², stand the 180-degree free-bend test without rupture,
and have an impact strength of 7.6 kg-m/cm². The white fumes of
zinc oxide given up in brass welding are harmful to the human
organism. Therefore, when an operator has to brass-weld for
a long time or inside brass vessels, he should wear a protective
mask or a respirator.
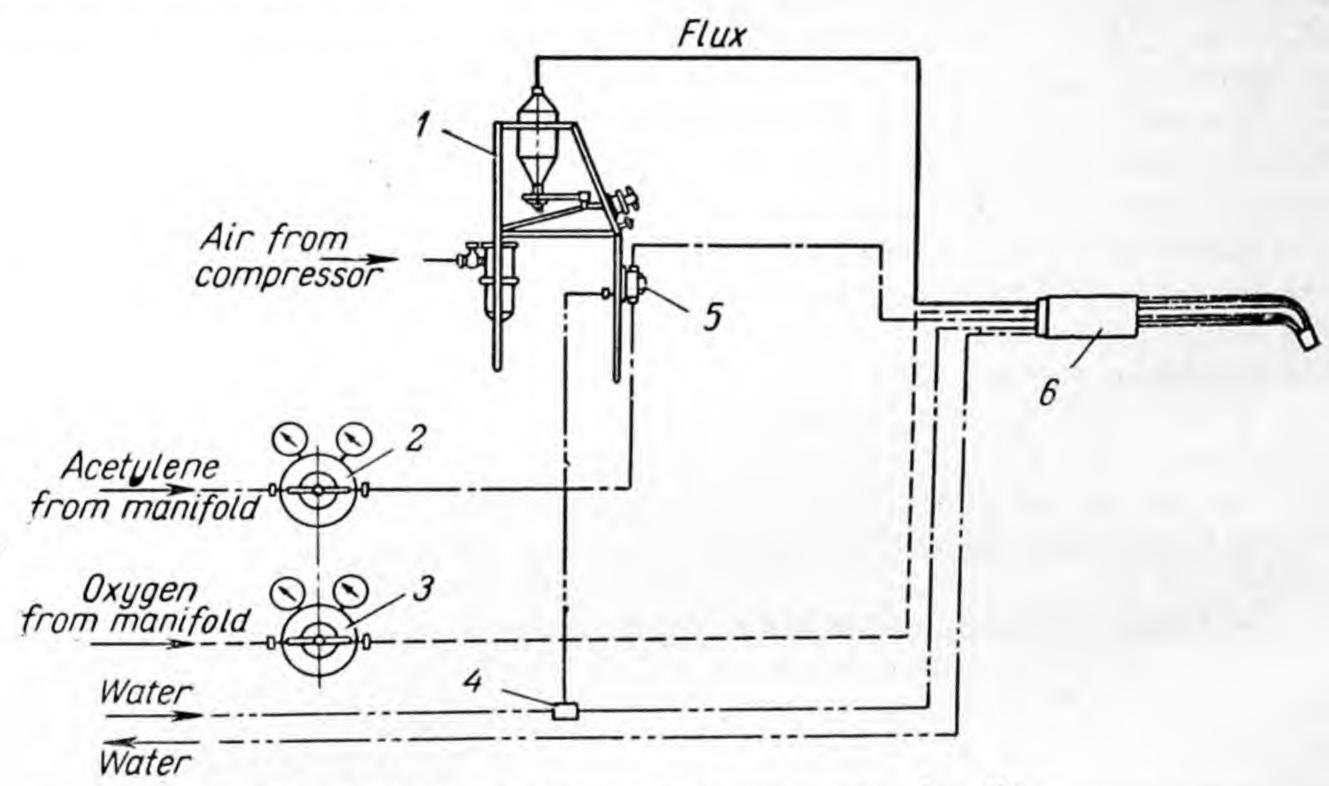
Fig. 111. A welding set-up using powder flux:
1 — flux dispenser; *2* — station acetylene regulator; *3* — station oxygen regu-
lator; *4* — tee-piece; *5* — shut-off valve; *6* — torch

For brass in heavy sections (from 40 mm up), the Autogenous
Welding Research Institute has developed a process based on the
gas flame and a powder flux. The process uses a special torch
into the flame of which a powdered flux is injected from a flux
hopper in metered quantities by a jet of nitrogen or dried air.
The torch is water-cooled and may therefore be used under the
most adverse conditions where the tip is heated to very elevated
temperatures. Fig. 111 shows a schematic arrangement of this
welding set-up. The flux is hydrated borax (calcined borax is
inconvenient to use as the flame blows it off the work). Flux
consumption is 2 grams per kilogram of melted metal. As applied
to brass ЛЖМЦ 40 mm thick, this process produces a dense and
well deoxidised weld metal with an ultimate tensile strength of
42 kg/mm² and an elongation of 34 per cent.

The Autogenous Welding Research Institute has also developed another process for welding brass alloys with a vapour flux. The flux is a boron-bearing liquid, designated БМ-1, which is a mixture of methanole (CH_3OH) and 55-70 per cent methyl
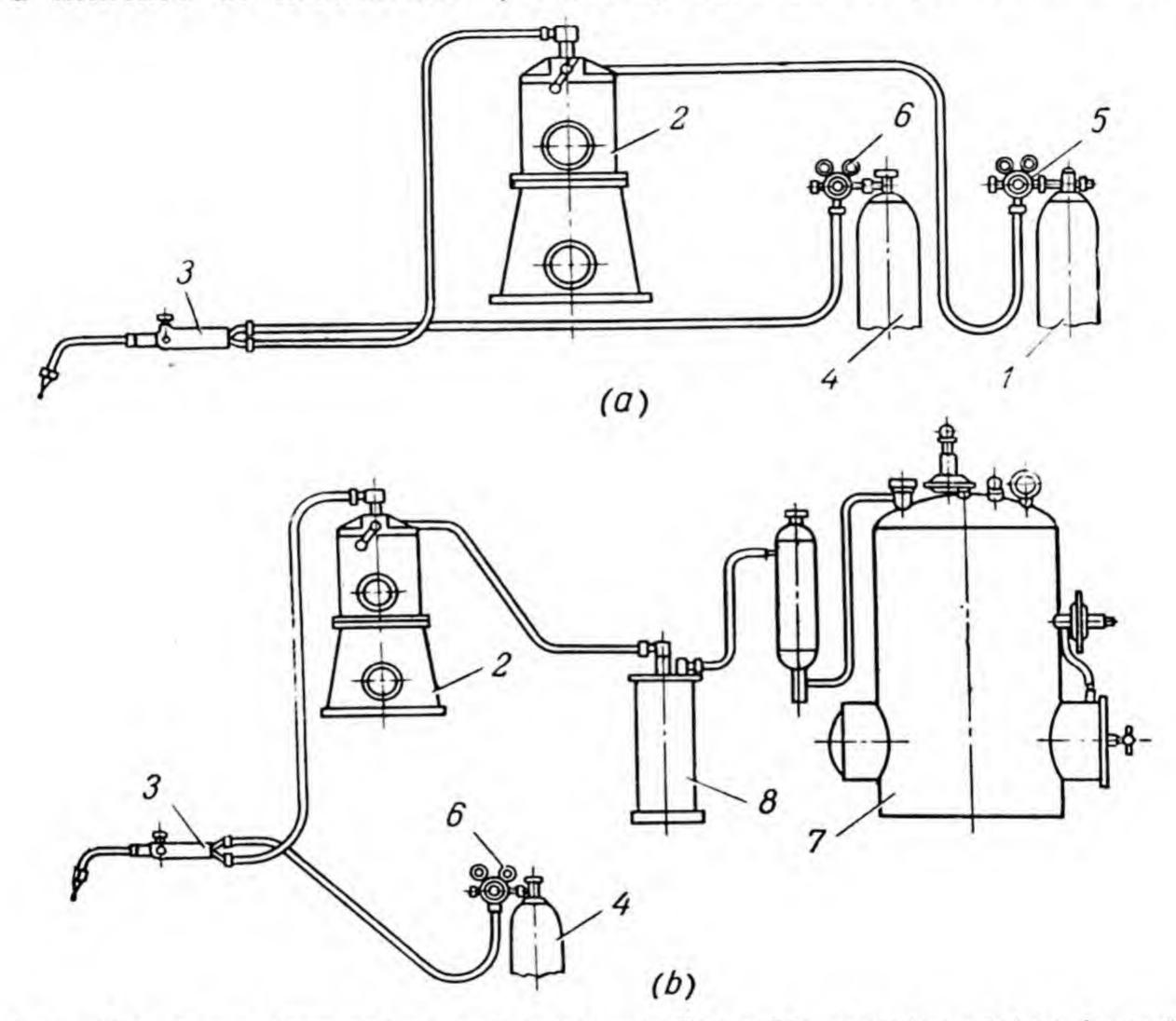
Fig. 112. A welding set-up using vapour flux with acetylene taken from (a) a cylinder and (b) a generator:
1 — acetylene cylinder; *2* — flux dispenser; *3* — torch; *4* — oxygen cylinder; *5* — acetylene regulator; *6* — oxygen regulator; *7* — acetylene generator; *8* — desiccator

borate, $B(CH_3O)_3$. The relevant set-up is shown diagrammatically in Fig. 112. Before it is admitted into the torch, the acetylene is bubbled through a liquid flux (which is the БМ-1 liquid) held in a flux dispenser where the acetylene picks up the flux in a vapour form. Carried by the acetylene into the flame, the БМ-1 vapour burns according to the reaction:

$$2B(CH_3O)_3 + 9O_2 = B_2O_3 + 6CO_2 + 9H_2O.$$

B_2O_3 is hydrated in the flame to form volatile boric acid (H_3BO_3) which is deposited on the workpiece and is again

decomposed to produce boric anhydride. Flux consumption is 70 grams of the БМ-1 liquid to the cubic metre of acetylene. This process raises welding rate and makes the welding operation healthy to the operator. The weld metal is dense and free of pores, and no peening is required. The mechanical properties of the weld metal are as follows: ultimate tensile strength, 38 kg/mm²; the bend test, 180°; impact strength, 15 kg-m/cm². This process is also used to brass-weld cast iron and to brass-face cast-iron and steel parts.

3. Bronzes

Bronzes are alloys of copper and variable percentages of tin, aluminium, manganese, silicon or some other metals. All bronzes may be classed into cast and wrought, depending on the way they are made into finished products.

In many applications copper-tin bronzes, using expensive tin, are successfully replaced by aluminium bronzes, manganese bronzes, silicon-manganese bronzes and other bronzes, all known as special bronzes.

Special bronzes are strong mechanically and resist corrosion efficiently.

Aluminium bronzes have found many uses in chemical engineering because of their high resistance to corrosion and heat. Aluminium and copper make several types of solid solutions. That with up to 9.4 per cent aluminium is a single-phase (or α-phase) alloy. As the aluminium content increases, another phase develops, increasing the strength and hardness of the alloy and reducing its percentage elongation. Aluminium bronzes have a greater coefficient of linear contraction (2.5 per cent) than copper-tin alloys (1.5 per cent). Additions of manganese, iron and nickel improve the mechanical, antifriction and plastic properties of copper-aluminium alloys. Aluminium bronzes like Бр. АЖ9-4, Бр. АМ10-3-7.5 are good substitutes for copper-tin bronzes such as Бр. ОЦ10-2, Бр. ОЦ8-4 and Бр. ОЦС6-6-3, and go to make mould castings and pipe fittings. Aluminium-iron-nickel bronzes like Бр. АЖН10-4-4, and Бр. АЖН11-6-6 posses high mechanical strength, resistance to wear and heat and are used instead of high-tin true bronzes like Бр. ОЦ10-2. Silicon-manganese bronzes (Бр. КМц3-1, also known as Everdur) are highly ductile, strong, resistant to corrosion, offer low friction, and possess high workability, sufficient castability and are readily welded by all methods. For this reason these alloys are extensively fabricated into equipment and fittings for the food-processing, chemical and some other industries.

The mechanical properties of some of these bronzes are summarised in Table 25.

Table 25

Mechanical Properties of Bronzes

Bronze	U. T. S., kg/mm²	Elongation, per cent
Бр. ОФ 6.5-0.4 (tin-phosphor) . . .	30^{+}	38^{+}
Бр. ОЦ 4-3 (tin-zinc)	30^{+}	38^{+}
Бр. А10 (aluminium)	$40\text{-}50^{++}$	$15\text{-}30^{++}$
Бр. АМц 9-2 (aluminium-manganese)	$50\text{-}60^{+}$	$30\text{-}35^{+}$
Бр. КМц 3-1 (silicon-manganese) . .	35^{++}	$14\text{-}20^{++}$
Бр. О10 (tin)	$25\text{-}30^{+++}$	$3\text{-}10^{+++}$

(+) Annealed strip
(++) Permanent-mould cast
(+++) Sand cast

Bronzes are gas-welded to repair parts or castings, to make rubbing surfaces wear-resistant and in some other cases. The main difficulty in bronze welding is the burn-out of the alloying elements, which fact usually results in a porous weld. When copper-tin bronzes are welded, the tin is segregated in the molten metal to form small globules of an alloy high in tin. At 1200° C, this alloy vaporises, producing fumes and a white coating of tin dioxide (SnO_2) around the weld. Tin dioxide readily dissolves in the molten weld metal and is difficult to flux out. In the case of aluminium bronzes, the main difficulty is that refractory aluminium oxide is formed which is difficult to withdraw from the weld metal (the melting point of the most commonly used bronzes usually lies anywhere between 1020° and 1060° C).

In welding silicon bronzes, a film of silica is formed over the weld pool, and the vaporisation of the other constituents (especially, zinc) is reduced. This, of course, improves the weldability of this type of copper-base alloys. Indeed, silicon bronzes possess the highest weldability of all bronzes.

When hot, cast bronzes are rather brittle. Therefore, when being welded, cast-bronze parts should be reliably clamped and guarded against shocks and jarring. Local heating may give rise to cracks due to locked-up stresses. To avoid this, it is good practice to give such parts a preheat of 450° C. A reducing flame should be employed, as the excess of oxygen produces oxide films of tin, aluminium and other elements. The oxide films usually form along the grain boundaries, thereby weakening the metal. The excess of acetylene in the torch flame results in a porous weld.

As in the case of brasses, the end of the inner cone of the flame should be held within 7 to 10 mm of the weld pool. When welding up through holes or cracks or where molten metal may escape, a mould of refractory clay should be laid up around the place to be welded, and graphite or steel back-up plates with a hollow for the sealing bead should be used.

The filler metal to use may be bronze rods 5 to 8 mm in diameter and 400 to 500 mm long, similar in chemical analysis to the parent metal. In the case of wrought bronzes, use is usually made of bronze wire or strip of the same grade as the parent metal. When cast rods are employed, they should be thoroughly cleaned of moulding sand, slag and other contaminants prior to welding. Preference should be given to chill-cast rods, as they have a cleaner surface. Tin bronzes are best welded with wire carrying 95-96 per cent copper, 3-4 per cent tin, 0.25 per cent phosphorus, i. e., close in chemical analysis to bronze Бр. ОФ6.5-0.4. The heat input should be adjusted so that the rate of acetylene flow is 125-175 lit/hr per millimetre of metal thickness.

In welding with concurrent heating, the rate of acetylene flow should be reduced to 100-150 lit/hr per millimetre of metal thickness, or the metal may be overheated and the resultant weld may be porous. The weld time should be kept to a minimum. The same types of joint preparation are employed as in the case of copper. Preference should be given to the single-V preparation with an included angle of 70-90°. Bronze welding requires the use of fluxes. These may be the same as for copper. For aluminium bronzes, the fluxes employed to weld aluminium alloys will do. One such flux contains 12-16 per cent sodium fluoride, 20 per cent sodium chloride, 20 per cent barium chloride, and potassium chloride in the remainder.

To help the withdrawal of the aluminium oxide (Al_2O_3) from the molten pool the latter should be continuously puddled with the welding rod. Tin and low-silicon bronze castings should be annealed at 450°-500° C after welding and quenched in water. Peening may be applied to wrought bronzes only. The gas welds in bronzes may have an ultimate tensile strength of up to 30 kg/mm².

4. Aluminium and Its Alloys

Aluminium has a low specific gravity (2.7) and a low melting point (657° C). Its coefficient of linear contraction is 1.7 per cent. Pure aluminium is a weak but highly ductile metal. For structural purposes use is made of a variety of aluminium alloys containing copper, manganese, magnesium, silicon and some other

elements. Aluminium and its alloys have a high thermal conductivity (0.52 cal/cm/sec/°C and high latent heat of fusion (100 cal/g). For this reason, aluminium welding requires high heat input. The melting point of aluminium alloys lies between 600° and 650° C.

Aluminium oxidises easily, forming a film of refractory oxide (Al_2O_3) which has a melting point of 2060° C and a specific gravity of 3.85, or greater than that of pure aluminium. This film protects the parent metal from further oxidation, but in welding it may persist in the molten metal as scattered inclusions. For a good weld to be obtained, it is vital to remove as much of the Al_2O_3 from the weld pool as possible.

Commercial aluminium alloys are classed into wrought alloys in which the amount of the alloying elements is below the saturation limit of solid solution at the eutectic temperature, and cast alloys which lie beyond the saturation limit. In turn, wrought aluminium alloys are classed into nonheat-treatable and heat-treatable. The line of demarcation between the two groups is the saturation limit of solid solution at room temperature. Some of the cast aluminium alloys can also be heat-treated to improve their properties.

The mechanical properties of aluminium and some of its alloys are presented in Table 26.

The welder has usually to deal with four principal groups of aluminium alloys. These are aluminium-manganese alloys of classification АМц, aluminium-magnesium alloys of classification АМг, aluminium-copper alloys of classification Д (duralumin) and aluminium-silicon alloys of classification AC (silumin). The first three groups are wrought alloys, and the fourth group embraces cast alloys. The alloys of classification АМц contain 1 to 1.6 per cent Mn and possess increased corrosion resistance, strength, hardness and good weldability. These alloys undergo practically no phase transformations during welding, as the manganese, instead of making solid solution in the aluminium, forms $MnAl_3$ or $MnAl_5$. At 657° C and 2.2 per cent Mn, the compound $MnAl_3$ makes with the aluminium an eutectic having the same melting temperature as aluminium. The АМг alloys contain 2-6 per cent manganese and have good weldability. The addition of some Mg to an alloy improves its resistance to corrosion and strength without impairing its ductility. The weldability of the alloy is decreased with increasing Mg content.

Duralumin-type alloys can be both strain-hardened and heat-treated by solution heat treatment followed by natural or artificial ageing. The solution heat treatment is based on the fact that some of the constituents have a higher solid solubility in alumin-
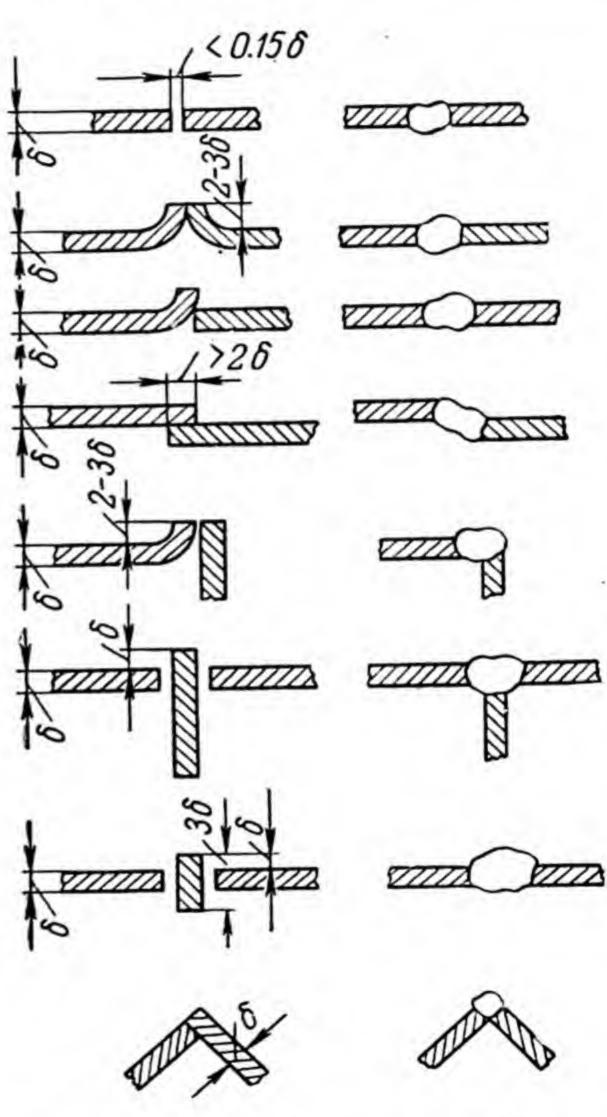
Table 26

Mechanical Properties of Aluminium and Its Alloys

Designation	Aluminium or alloy	Temper	U. T. S., kg/mm²	Elongation, %
A1	Aluminium, commercial, wrought	Soft	8-10	32-40
A2	Ditto	Hard	15-25	4-8
A3	Ditto	as-cast	9-12	11-25
АМц	Aluminium-manganese alloy, wrought	Annealed (M)	13	20
		Semicold-worked (П)	16	10
АМг	Aluminium-manganese alloy, wrought (magnalium)	Annealed (M)	20	23
		Semicold-worked (П)	25	6
АМг5	Ditto	Annealed	27	23
		Cold-worked	33-36	12-20
Д16	Duralumin, wrought, heat-treatable	Annealed	18-20	12-18
		Solution heat-treated and then artificially aged	46	17
АВ	Avial, wrought, heat-treatable	Annealed	11-14.5	24
		Solution heat-treated	22	22
		Solution heat-treated and then naturally aged	33	12
АЛ4	Silumin, cast, inoculated with sodium fluoride	Sand-cast, solution heat-treated and then artificially aged	26	40

ium at elevated temperatures than at room temperature (Al-Cu alloys). Accordingly, the alloy is heated to a temperature at which the hardening constituents go into solution, and then quenched to keep these constituents in solution. The as-quenched alloy is then in a state of supersaturation. The hardening constituents will precipitate out with time at room temperature, and the alloy will gain in strength and hardness, but lose some of its ductility. The hardening effect may be fully removed by annealing.

Cast aluminium-silicon alloys are not heat-treatable. Instead, their mechanical properties can be improved by alloying with what are called modifiers. The addition of copper and magnesium to silumin makes the alloy heat-treatable. Of the cast-aluminium alloys, the one that has the highest mechanical properties is magnalium which contains 10 per cent Mg and can be heat-treated.

Fig. 113. Edge preparation and joint designs for light-gauge aluminium alloys without filler metal

Sheet aluminium or aluminium alloys up to 4 mm thick can be square butt-welded, with the edges spaced 0.5 mm apart. In the case of thicknesses from 4 to 18 mm, the V-preparation is resorted to, while on sheet over 18 mm thick the X-preparation with a bevel angle of 30-35 degrees should be employed. The root face should be one-fourth of the sheet thickness. Thin sheets (under 2.5 mm) may be welded either without filler metal with the edges flanged, or with a strip placed between the edges (Fig. 113). Lap joints should be avoided, as flux or slag may find its way between the laps to cause heavy corrosion afterwards.

16*

Prior to welding, both the edges and welding rod, if used, should be degreased by some solvent and then wire-brushed to remove aluminium oxide within 30 to 60 mm on either side of the joint. The welding operation should be commenced not later than 2 hours after the cleaning operation, as the surfaces may be again oxidised.

Mechanical cleaning cannot remove the oxide coating uniformly, and the welding quality may therefore be rather low. Better results are obtained by chemical cleaning. Alloys of classifications АМц and АМг5 may be degreased with alkaline solutions (150 grams of caustic soda to one litre of water, a holding time of 10 min at 70° C) and then treated for two minutes with a 25-per cent solution of orthophosphoric acid. Alloys of classifications АД1 and АМг can be degreased with a similar alkaline solution and then pickled with a 30 or 35-per cent solution of nitric acid for one minute. After each operation, the edges should be thoroughly rinsed with water, wiped with a piece of cloth, and allowed to dry. Thus treated, the metal should be welded not later than 8 hours after pickling, before the edges oxidise again.

In welding, the aluminium is melted quickly, while the oxide coating remains unmelted, and it is difficult for the operator to notice when the pure aluminium begins to melt under the oxide film, which fact calls for utmost care on his part. The heat input should not be excessive so as not to burn the metal through. The relationship between the metal thickness and the flame output is as follows:

Metal thickness, mm	0.5-0.8	1	1.2	1.5-2	3-4
Rate of acetylene flow, lit/hr	50	75	75-100	150-300	300-500

For heavier sections, the rate of acetylene flow may be 110-130 lit/hr per millimetre of metal thickness.

The oxide coating is removed and withdrawn from the weld pool by means of suitable fluxes which may be in powder or paste form (see Chapter X, Sec. 2).

After welding, surplus flux should be thoroughly removed by rinsing the part with hot water to prevent corrosion. Surplus flux can be efficiently removed by rinsing the weld with warm water, then treating it with a 5-per cent solution of nitric acid with an addition of 2-per cent solution of potassium bichromate for 5 minutes, then again rinsing it with water and allowing it to dry. As a result, a protective oxide coating is again formed over the weld and the adjoining areas.

The filler metal to use is wire or strip of the same chemical analysis as the parent metal. In welding alloys of classification AMц and heat-treatable alloys, good results are obtained with wire of AK alloy containing 5 per cent silicon, for it increases the fluidity of the filler metal and reduces its shrinkage on cooling.

On the other hand, AK wire should not be used to weld non-heat-treatable alloys of classifications AMr and AMr5, as inter-crystalline corrosion may occur in the weld metal in service. In addition, the weld becomes coarse-grained and loses some of its strength. Better results are obtained with AMr wire with an excess of magnesium so as to bring down the melting point of filler metal.

Cast aluminium alloys are successfully welded with aluminium, AMц or AK wire. The diameter of the wire should be as follows:

Metal thickness, mm . .	1.5	1.5-3	3-5	5-7	7-10
Wire diameter, mm . .	1.5-2	2.5-3	3-4	4-4.5	4.5-5.5

Either a normal or a slightly carburising flame should be employed. Preference should be given to the leftward (forehand) technique so as to avoid overheating the metal and the excessive grain growth. On sections under 3 mm thick, the flame and the rod should be advanced straightly forward. The welding operation should be carried on without holding the torch for any appreciable time at one place, so that the torch tip and the filler rod make an angle of 20-45° and 40-60° with the part, respectively. The weld should be commenced within 80-100 mm of the end of the joint, and the unwelded length should be welded afterwards in a backward direction. After welding, the seam may be slightly peened while cold.

The gas welds made in aluminium alloys have an ultimate tensile strength of 8-11 kg/mm² and a bending angle of 110-180°. In the case of AMц aluminium alloy sheet 10-16 mm thick, the ultimate tensile strength of the weld is 8.5-11.5 kg/mm², the bending angle is 113-180°, and the impact strength is 2-3 kg-m/cm². For comparison, the impact strength of the parent AMц metal is 5.5-7.5 kg-m/cm².

Aluminium castings have to be welded with a preheat of 250-260° C. Cracks are welded up in increments of 50-60 mm, working from the centre out towards the opposite ends.

If the weld metal is to have a fine-grained structure and to be free from internal stresses, it is a good plan to anneal weldments from cast aluminium alloys at 300-350° C and then to cool slowly.

5. Magnesium Alloys

Alloying magnesium with manganese, zinc and aluminium produces alloys which have found many commercial applications. These alloys have a low specific gravity (1.76-1.8 g/cm^2) and fairly good mechanical properties (an ultimate tensile strength of 21-34 kg/mm^2 and an elongation of 8-20 per cent). Their coefficient of thermal conductivity lies between 0.18 and 0.35 cal/cm/sec/° C, and their coefficient of linear expansion is 26×10^{-6}. Pure magnesium melts at 650° C, while the melting point of its alloys is anywhere from 460° to 650° C. The mechanical properties of cast magnesium alloys such as МЛ-4, МЛ-5 and МЛ-6 containing 5-11 per cent Al, up to 3 per cent Zn and 0.1-0.5 per cent Mn, can be improved by solution heat treatment and ageing, while those of МЛ-2 (1-2 per cent manganese, the remainder magnesium) and МЛ-3 (2.5-3.5 per cent Al, 0.5-1.5 per cent Zn, 0.15-0.5 per cent Mn, the remainder magnesium) cannot. Magnesium readily combines with oxygen to form a coating of oxide, MgO, less strong than aluminium oxide and therefore unable to protect the parent metal from corrosion efficiently. An addition of manganese to a magnesium alloy improves its corrosion resistance and refines its structure. The chemical analyses and weldability of some of the magnesium alloys are tabulated in Table 27.

The principal difficulty involved in the gas welding of magnesium alloys is their inflammability. Their welding is further hampered by their low melting point and by the formation of the refractory surface oxide (2500° C). Therefore, the operator has to exercise special care so as not to burn through the metal. In the case of binary Mg-Mn alloys (such as MA1 and MA8), heating brings about grain growth in the weld metal and the heat-affected zone, and the strength of the alloy is sharply reduced. Magnesium alloys containing 0.2-0.4 per cent titanium or about 0.5 per cent selenium are less susceptible to grain growth due to heating.

In addition to their intensive oxidation, magnesium alloys, when heated to near their melting point, eagerly combine with nitrogen to form magnesium nitride. Magnesium also reduces water vapour by forming MgO and producing hydrogen. The latter dissolves in the molten metal and may result in a porous weld. The presence of MgO and magnesium nitrides in the weld metal impairs its strength and makes it brittle. Mg-Al alloys are susceptible to microscopic porosity due to the formation of intradendrite shrinkage cavities. Mg-Zn alloys are red-short and may crack during welding much as ternary alloys do.

Table 27

Magnesium-base Alloys

Desig-nation	Chemical analysis, %	Form and condition	Gas weldability
MA1	0.3 Al, 1.3-2.5 Mn, remainder Mg	Sheet, strip, rods, forgings, pressings; all annealed	Limited
MA2	3-4 Al, 0.15-0.5 Mn, 0.2-0.8 Zn, remainder Mg	Extruded bars and rods, forgings, pressings	Satisfactory
MA3	5.5-7 Al, 0.15-0.5 Mn, 1.5 Zn, remainder Mg	Ditto	Ditto
MA5	7.8-9.2 Al, 0.15-0.5 Mn, 0.2-0.8 Zn, remainder Mg	Ditto, solution heat-treated	Ditto, but worse than of MA3
MA8	0.1 Al, 1.5-2.5 Mn, 0.3 Si, 0.15-0.25 Ce, remainder Mg	Sheet, strip, annealed	Satisfactory
МЛ4	5-4 Al, 0.15-0.5 Mn, 2-3 Zn, 0.25 Si, remainder Mg	Sand-cast, solution heat-treated	Limited
МЛ5	7.5-9.3 Al, 0.15-0.5 Mn, 0.25 Si, 0.2-0.8 Zn, remainder Mg	Solution heat-treated and aged	Ditto

The weldability of magnesium alloys depends to a great extent on their content of manganese and, especially, zinc. The MA1 alloy, containing a fairly large amount of manganese, possesses poor weldability because of its very narrow solidification range and high melting point. Alloys high in zinc tend to be red-short and may develop cracks in the process of welding. Increased aluminium moderates the effect of zinc. In welding magnesium alloys, care must be taken to advance the torch as quickly as possible so as to provide for the sufficient cooling rate and to prevent cracking thereby. Cast magnesium alloys are more difficult to weld than wrought alloys, as they tend to produce porous welds, coarse-grained structure and oxide inclusions in the weld. Better results are obtained with single-pass welds.

On thicknesses under 2 mm magnesium alloys can be gas-welded without joint preparation. On heavier sections, the V-pre-paration is used, with an included angle of 80-90 degrees, and with no gap left at the root. The root face should be not less than 1.5 mm. Prior to welding, the edges must be thoroughly de-greased and wire-brushed. Better results are obtained with chemical cleaning, when the edges are degreased with a 5-per cent solution of NaOH for 10 minutes at 65° and then washed first with hot and then cold water. After degreasing, the metal is pickled for two minutes with an alkaline solution (200 grams of chromic anhydride, 30 grams of sodium nitrate, 5 grams of potassium fluoride, and 1,000 cm³ of water), washed with hot and cold water, and rubbed with a piece of cloth in a water bath. As a rule, magnesium alloys are given a corrosion resistant chrome coating by the producer. If this is the case, this protective coat-ing should also be removed prior to welding by pickling the metal in a 18-per cent solution of chromic acid at 90-100° C for 5 minutes.

The heat input should be so adjusted that the rate of acety-lene flow is 150 lit/hr on thicknesses of 1 or 2 mm and 300 lit/hr on thicknesses of 5 or 6 mm, and have a slight excess of acety-lene. The angle between the flame and the work should be kept to a minimum (about 10°), with the inner cone held within 1.5 to 3 mm of the weld pool. The welding rod should never be allowed to dip into the pool as this may contaminate the weld with oxides.

Magnesium alloys should preferably be joined by butt, tee and corner joints. Lap joints should be avoided, as with them the metal may be easily overheated and develop cracks. The welding speed should be maintained as high as possible. Usually, for a thickness of 1 or 2 mm, the welding speed should be 6 to 9 m/hr, and for a thickness of 5 mm or more, 3 or 4 m/hr. Seams should be completed in a single pass with the leftward (forehand) tech-nique only without manipulating the torch in a criss-cross (weave) manner.

The filler metal may be wire of the same chemical analysis as the parent metal.

The use of flux is most important in the welding of magne-sium alloys. This may be a mixture of 31 per cent lithium fluo-ride, 14 per cent magnesium fluoride, 7 per cent calcium fluoride, 15 per cent barium fluoride and 33 per cent aluminium fluoride. Good results are usually obtained with the flux consisting of 40 per cent lithium chloride, 40 per cent sodium chloride, and 20 per cent calcium fluoride. Use may also be made of the fluxes for aluminium alloys.

The flux is applied to the welding rod and on either side of the joint, close to the edges. After welding, surplus flux should be removed immediately by washing the weld with hot water.

To protect the weld and adjacent metal from corrosion, it is good practice to apply a protective coating by dipping the work for 30 minutes in a boiling solution of the following composition (by weight): 3 per cent ammonium sulphate, 1.5 per cent ammonium bichromate and 0.35 per cent ammonia (sp. gr. 0.88).

For better ductility, the welds in magnesium alloys should preferably be peened. In the case of heavy thicknesses and castings, it is a sound practice to use a preheat of 300-350° C.

The post-welding stresses in magnesium alloys may be relieved as follows:

	Heating, °C	Hold, min
Annealed alloys MA1 and MA3 . .	260	15
Cold-worked alloys MA1 and MA8	205	60
Cold-worked alloy MA3	190	60
Solution heat-treated alloy MA5 . .	315	15
Solution heat-treated and aged alloy MA5	205	60

The gas welds made in magnesium alloys have 60 to 80 per cent of the strength of the parent metal.

6. Nickel and Its Alloys

Nickel and its alloys are widely used commercially because of their excellent physical and chemical properties: ductility, malleability, chemical stability, etc. Commercially pure nickel contains 97.6 to 99.8 per cent nickel. Cold-drawn nickel has an ultimate tensile strength of 80-90 kg/mm², while annealed nickel has an ultimate tensile strength of 45-52 kg/mm² and an elongation of 35-40 per cent. Nickel melts at 1452° C, can be hot-worked at 1100-1200° C and annealed at 780-850° C. Its linear shrinkage is 1 per cent. Sulphur is the most harmful impurity in nickel, and its content should not exceed 0.005 per cent in Grade H0 and 0.01 per cent in Grade H1. Lead, even in negligible amounts (a few thousandths of one per cent), makes nickel red-short as it is absolutely unsoluble in nickel. Nickel is present in a great variety of alloys to which it imparts high resistance to corrosion and good physical and mechanical properties.

The most commonly used of them are ТП and ТБ copper-nickel alloys (nickel bronzes) containing up to 6 per cent Ni and

the remainder copper; nickel bronzes containing 20 to 43.5 per cent Ni; Monel metal containing 28 per cent Cu, 68 per cent Ni, 1.5 per cent Mn and 2.5 per cent Fe and resisting oxidation efficiently up to 750° C; Ni-Cr alloys; nickel-copper-zinc alloys (nickel silver); and a range of special acid-resistant alloys containing nickel, molybdenum and iron. Incidentally, Monel metal is more resistant than pure nickel to some of the corrosive agents (such as salt solutions, steam at 750° C, organic acids, hydrochloric and phosphoric acids), for which reason it is widely used in chemical and food-processing equipment.

The welding of nickel runs into the following difficulties:

1. The molten metal absorbs gases, but their solubility sharply drops when the deposited metal solidifies, and so the resultant weld may develop porosity. To reduce its probability, it will be a good plan to employ the rightward (backhand) technique of welding, as with it the rate of cooling is slowed down.

2. A refractory coating of nickel oxide with a melting point of 1650-1660° C forms over the weld pool. Its removal calls for the use of fluxes which may be roasted borax, a mixture of 25 per cent borax and 75 per cent boric acid, a saturated solution of boric acid in alcohol, a mixture of 50 per cent boric acid, 30 per cent borax, 10 per cent common salt, and 10 per cent barium carbonate. In the more complex formulae, in addition to borax and boric acid, use is made of chlorides of magnesium, manganese, lithium and cobalt, as well as ferrovanadium and titanium concentrate.

Nickel is welded by the gas process fairly well. Sheets under 1.5 mm thick are welded without any filler metal, with the edges flanged so that the flanges are one or one-and-a-half times the thickness of the work. Sheets up to 4 mm thick are joined by means of square butt welds. On heavier gauges, the edges are V'ed to an angle of 35 or 45 degrees. Lap joints are not used because of the appreciable warpage of the work due to heating. Prior to welding, the parts to be joined should be tack-welded at every 100 to 200 mm. The weld is made in increments, using the back-step sequence. There should be no excess of oxygen in the flame, as this may produce porosity, and the deposited metal will be brittle. The flame may be slightly carburising, with the rate of acetylene flow adjusted at 140-200 lit/hr per millimetre of metal thickness for nickel, and 100 lit/hr per mm of metal thickness for Monel metal. The filler material may be in wire or strip form of the same analysis as the parent metal. The wire diameter should be half the thickness of the work. Good results have been obtained with nickel wire containing up to 2 per cent manganese and not over 0.2 per cent silicon. The resultant

weld has an ultimate tensile strength of 26-28 kg/mm² and withstands a bending test through an angle of up to 90 degrees.

The welding of Ni-Cr alloy (75-80 per cent Ni, 15-18 per cent Cr and 1.2-1.4 per cent Mn), which has a melting point of 1390° C and low thermal conductivity, is handicapped by the fact that the molten pool is blanketed by a coating of refractory chromium oxide which is usually removed mechanically. The welding operation should be carried on as quickly as possible and with as few interruptions as practical, for multipass welding and rewelding give rise to cracks, grain growth and intercrystalline corrosion in the weld. The flame should have an excess of acetylene and be adjusted so that the rate of acetylene flow is 50-70 lit/hr per millimetre of metal thickness. The flux paste used consists of 40 per cent borax, 50 per cent boric acid and 10 per cent of sodium chloride or potassium fluoride. The paste is diluted with water. The filler material is supplied in a strip 3 or 4 mm wide of the metal being welded or Ni-Cr wire of classification ЭХН-80. After relief-anneal, the weld has an ultimate tensile strength of 35-45 kg/mm².

7. Lead

Lead melts at 327° C and, when heated, easily oxidises to form a coating of lead oxide (PbO) which has a melting point of 850° C.

As it is chemically stable, lead is widely employed in chemical equipment which is to be exposed to sulphuric acid. The acid acts upon the lead to form a protective coating of lead sulphide which is resistant to sulphuric acid with a concentration of up to 80 per cent. Lead is likewise resistant to other (sulphurous, fluoric and phosphoric) acids capable of forming hardly soluble compounds on its surface. On the other hand, lead is easily attacked by the agents which do not form protective coatings or which produce easily soluble compounds with lead. This applies to water containing carbon dioxide, as the latter forms lead carbonate and bicarbonate which dissolve in water without difficulty.

Lead can be welded or, as is sometimes inaccurately described, "burned" with the oxy-hydrogen, air-acetylene or oxy-acetylene flame. Sheets up to 1.5 mm thick are flange-welded without any filler metal. Sheets up to 6 mm thick are square butt-welded, while on heavier thicknesses the edges are bevelled to an angle of 30 or 35 degrees.

Lead may also be joined by lap welds. The flame should not have an excess of oxygen and should be adjusted so that the rate

of acetylene flow is 15-20 lit/hr per millimetre of metal thickness. The torch angle should be kept as flat as possible and the flame should be advanced as quickly as possible. For a thickness of 3 or 4 mm, the welding speed should be 6.5 to 8 m/hr. The "burning bar", as the filler rod is termed in the case of lead welding, may be lead wire the diameter of which should be two or two-and-a-half times the thickness of the work. Lead is often welded by what is called the fish-scale technique. In this case, the operator heats both the edges of the work and the filler rod to the near-fusion point, then withdraws the torch, and allows the drop of metal to break away from the tip of the rod and fall onto the melted edges, where the two metals mix and form the weld metal. Every next drop should overlap the previous one by at least half the length. On gauges over 8 mm, multipass welding is employed. Lead can be welded in the downhand, vertical and overhead positions. In the case of lap joints, the top lap is slightly bent away and used as the filler material. The fusion of the metal and the removal of PbO is greatly facilitated by the use of a flux consisting of equal parts of rosin and stearine.

GAS-PRESSURE WELDING

1. Basic Features

In gas-pressure welding, the ends of two pieces of metal to be welded are heated by a multiflame torch to a welding heat and are then pressed together by an axial force.

There are two basic types of gas-pressure welding: (a) with the edges brought up to a pasty state (closed-butt or solid-phase welding) and (b) with the edges brought up to the melting point (open-butt welding). The sequence of operations for both types is diagrammatically shown in Fig. 114.

In the case of solid-phase welding, a weld may be completed by two methods. One of them, called the constant-pressure method, consists in that the parts to be welded are pressed together by a constant axial force which is removed after the desired amount of upset has been attained. By the other one, called the set-temperature method, the parts to be welded are first pressed together by a small force, then are heated to the set temperature, and the axial force is raised to the maximum; the weld is completed when the requisite amount of upset is attained.

In open-butt welding, the flame is made to play on the ends of the parts. After they have been raised to the fusion point, the ends are compressed together to complete the weld, and molten slag is expelled. The ends can be heated sideways (at *I* in Fig. 114), when the flame is applied from the outside all the way round the perimeter of the workpiece, or between the end faces (at *II* and *III* in Fig. 114).

The heat input should be greater for the open-butt than for the closed-butt method, but in the former case the edges need not be machined or prepared for welding. The flush at the joint may be removed, if necessary, by filing, machining, grinding or forging. After peening, the weld may be heat-treated with the same torch as it has been welded in order to refine the structure of the weld metal.

The gas-pressure process is employed to weld steam, gas and liquid pipelines, as well as parts for locomotives and railway car-

riages. Its advantages are that the process is very simple to carry out, its equipment is inexpensive to install, small in weight and size, there is no need in heavy electric sources, and, consequently, gas welding units are, in fact, self-contained.
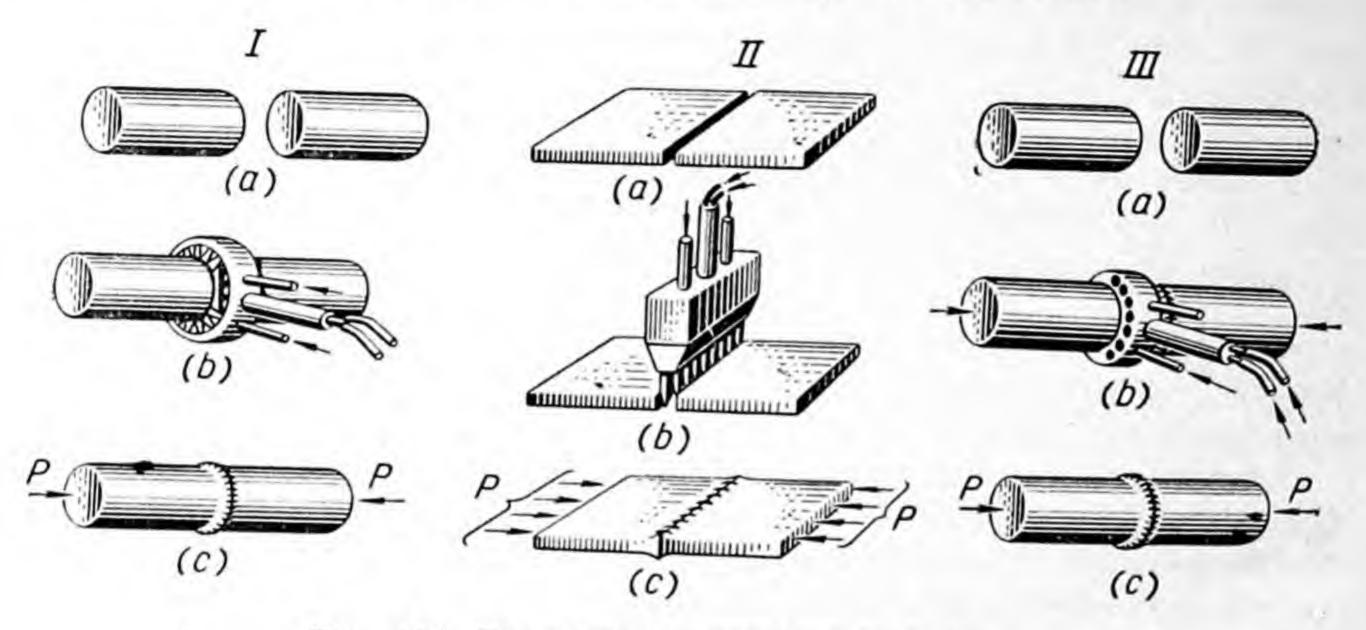
Fig. 114. Gas-pressure welding techniques:
I — round-bar welding with a ring multiflame torch:
a — bars prior to welding; *b* — heating the bars (in the case of the closed-butt method, the ends are brought together under pressure and the torch is caused to play over the joint area; in the case of the open-butt method, the ends are spaced the desired distance apart, while the flames are made to play on the ends through the gap until the ends are brought to the fusing point); *c* — the ends are forced together to complete the weld;
II — sheet welding with a straight-line torch by the open- and closed-butt methods (see *a*, *b* and *c* in *I*);
III — bar welding by the open-butt method (*a* and *c* same as in *I*; in *b* the ends are brought to the fusion point)

In the U.S.S.R., a broad range of simple and reliable units have been developed for gas-pressure welding, capable of joining parts of large cross-section which are difficult to weld by any other process. An operator can acquire the knack of running these units in a matter of a few days, the more so that mechanisation has greatly simplified his task.

The quality of a gas-pressure weld is a function of many variables including the temperature of heating, the pressure applied, the amount of upset, the heat input and nature of the flame, the swing of the torch across the workpiece, etc. The most important factors are the temperature of heating and the pressure applied. If the parts are heated to a low temperature and the pressure applied to compress them is insufficient, not all of the oxide coating will be removed from the crystallites, and the weld will be weak, possessing low resistance especially to shock and

vibration loads. For a sound gas-pressure weld to be obtained, the heating temperature should be above the lower limit of weldability for a given steel but below the point at which the steel may be burned. For plain low-carbon steel, the heating temperature lies anywhere between 1180° and 1260° C. The pressure should be 1.5-2.5 kg/mm² for solid low-carbon steel and 2.0-3.5 kg/mm² for low-carbon pipes. For pipes from alloy steels, some investigators suggest a pressure of 5-6 kg/mm², and for pipes from chrome-nickel austenitic steel, such as 1X18H9T, 10 to 12 kg/mm².

The amount of upset should be 0.3 d for solid rounds, and (1-1.3) S for pipes, where d is the diameter of a round, and S is the thickness of a pipe wall. If the upset is too small, the surface oxide will not be broken down, nor will all of the oxidised metal be removed from the welding zone. Both factors reduce the strength of the welded joint.

The specific rate of acetylene flow which provides for the adequate heating of the ends of the parts to be welded is 1-2.5 lit/hr/mm² for solid rounds and 1.8-2.2 lit/hr/mm² for pipes. The flame should have 5 to 8 per cent excess acetylene so as to exclude the possibility of the metal at the joint being oxidised. This is necessitated by the fact that in gas-pressure welding the inner cone is rather remote from the metal surface, and the reducing atmosphere can only be obtained by providing an excess of acetylene in the flame. This is adjusted by eye and is judged by the presence of a greenish halo around the tip of the inner cone.

When properly made, gas-pressure welds are very strong and ductile. This is because the workpiece is heated gradually and uniformly, with the gas envelope of the flame reliably shielding the metal from oxidation by the oxygen of the air, and the resultant weld metal is uniform in structure. The subsequent machining and heat treatment add to the strength and ductility of the weld still more. In fact, gas-pressure welds are infrequently superior to the parent metal in tensile strength, while their ductility, after peening and normalising, is comparable with that of the parent metal. Without normalising, the impact strength is reduced to about 20 per cent of that of the parent metal prior to welding. After normalising, the impact strength of the welded specimen is raised to 80 per cent of that of the parent metal prior to welding. Some of the reduction in the impact strength is probably caused by the grain growth due to heating and the duration of the welding operation. The endurance limit is almost the same for both welded specimens and the parent metal.

The weld metal is in the most cases uniform in microstructure. If, however, pressure, weld time and heating temperature have been unhappily matched, a ferrite streak of light colour may be noticed in the weld. Among the causes contributing to this ferrite streak some of the investigators suspect decarburisation, insufficient diffusion of oxygen from the weld into the parent metal and of carbon from the parent metal into the weld, the preferential precipitation of ferrite around the crystallising nuclei present in the welding zone, and, as a corollary, incomplete recrystallisation in the welding zone. When the welding variables are chosen properly, no ferrite, streak appears in the weld. It
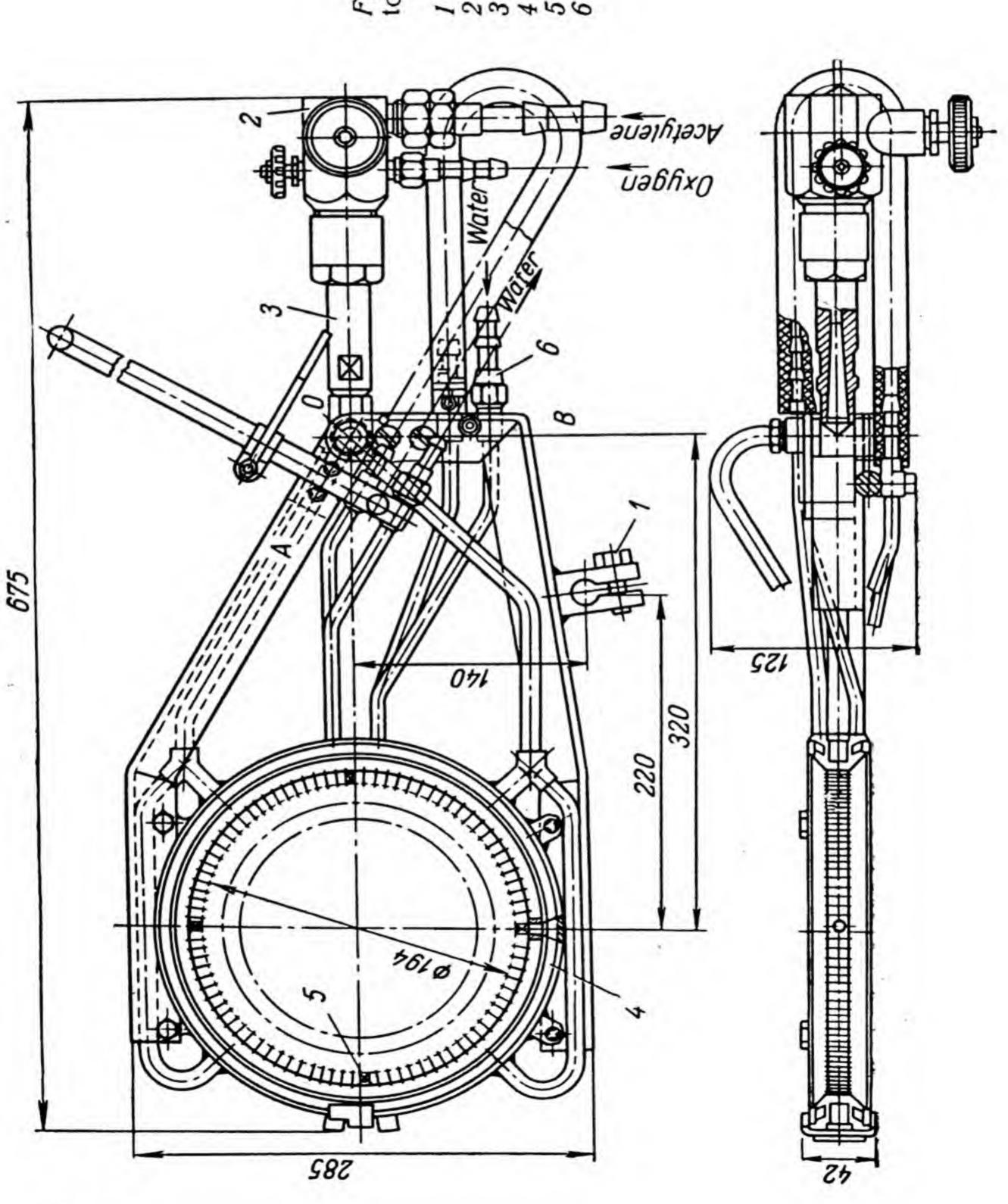
Table 28

Technical Data on Gas-pressure Welding Units

Welding variable	Machine designation			
	СГП-3р	СГП-1р	СГП-7	МГПС 15/160-53
Max. upset force, kg	3,000	13,000	12,000	15,000
Max. clamp force, kg	6,000	25,000	24,000	42,000 for V-slot jaws and 30,000 for jaws fitting workpiece
Max. upset, mm	16	50	50	40
Max. torch travel, mm	40	100	100	100
Max. dia. to be welded, mm				
tubes	50	cross-sectional area of weld, 6,000 mm²	160	30-160
rods	35		100	30-60
Clamp spacing, mm . . .	150	330	300	—
Upset drive	manual	pneumatic	pneumatic	pneumatic
Air line pressure, atm (gauge)	—	5	4-6	5-6
Clamping	manual	manual	pneumatic	pneumatic
Torch drive	manual	manual	power	manual/power
Overall dimensions, mm	800 × 400 × × 650	1,415 × 810 × × 1,352	1,630 × 1,150 × × 1,830	2,025 × 975 × × 1,300
Weight, kg	219	890	2,000	1,380

Fig. 115. Type МГ-50 ring torch for gas-pressure welding:

1 — clamping bolt;
2 — shank;
3 — mixer chamber;
4 — ring head;
5 — tips;
6 — inlet nipple for cooling water

675

285

Ø194

140

220

320

125

42

Acetylene

Oxygen

Water

Water

17 — 413

should be noted, though, that the presence of a ferrite streak does not affect the tensile and impact strength of the welded joint to any appreciable degree. Other investigators believe that the ferrite streak results from the reduction of the iron oxides at the joint by the carbon of the flame. Accordingly, they recommend that, if the ferrite streak is to be avoided, the joint should first be sweated all the way round its perimeter and then raised to the welding heat.

A typical gas-pressure welding unit consists essentially of a press complete with clamping and upset facilities, a split-type ring burner, and a gas supply system. The clamps can take bars, rods, strips, pipes and tubes. The burner is of multitip construction and is mounted in the arm of the press. The upset facility supplies the requisite axial force. Both clamping and upset pressure can be provided either (a) manually, or (b) pneumatically, or (c) hydraulically. Hand-operated presses are employed to gaspressure-weld bars and rods not over 1,000 sq mm in cross-section. The clamp force should be approximately twice the upset force.

Table 28 presents technical data on some of the gas-pressure welding units available commercially in the Soviet Union.

Each half of the split-type multitip ring burner used for gaspressure welding has a water-cooling system of its own. Burners may have either one or two shanks, depending on design and gas flow rate. The size and shape of a burner vary with the workpieces to be welded, as all burners for gas-pressure welding are of special-purpose construction.

Fig. 115 shows one of the most commonly used burners, the Type МГ-50, which has one shank supplying gas to both halves of the burner head. МГ burners heat and weld pipes with an O. D. of 30 to 170 mm and a wall thickness of 3-14 mm, as well as round bars and rods from 30 to 120 mm in diameter.

2. Procedure

(1) **Closed-butt or Solid-phase Welding.** (a) _Edge Preparation._ The edges should be bevelled to an angle sufficient for the uniform heating of the joint across its entire section. Usually, the angle of bevel is 6 to 15 degrees. In the case of pipes with a wall thickness of 3.5 to 8 mm, the best results have been obtained with a bevel of 10 to 15 degrees, as the penetration is uniform throughout the thickness of the pipe wall and no appreciable flash is formed on the inside.

The fusion faces should be thoroughly cleaned of rust, mill scale, grease, paint and other contaminants. The members to be

joined should be lined up as accurately as possible. The offset of the edges should in no case exceed 1 to 1.5 mm.

(*b*) *The Welding Flame.* The flame should have a small excess (up to 8 per cent) of acetylene. An oxidising flame should be avoided, as it produces oxides in the weld, thereby weakening the strength of the joint. On the other hand, a flame with an appreciable excess of acetylene will carburise the weld metal, making the latter more brittle than the parent metal.

The workpiece should be so positioned in the burner that the inner cones are the same distance from the metal surface. Practically, this distance should be 8 to 20 mm. The burner should heat the joint uniformly all the way round the circumference.

As a rule, the lower side of the joint will be heated more than the upper one. To avoid this, the burner should be slightly offset downwards, so that the inner cones on the top are 1 to 2 mm closer to the surface than those on the underside.

(*c*) *The Welding Conditions.* In the case of gas- pressure welding, the welding variables include the rate of acetylene flow in litres per hour per square centimetre of joint area; the specific clamp pressure in kilograms per square centimeter, and the temperature of the joint. They have all been summarised in Sec. 1.

(*d*) *The Welding Technique.* The members are securely clamped in the press and brought together. The axial upset is adjusted. The burner is fired, the composition of the flame is checked, the flame position with respect to the workpiece is verified, and the heat is applied to the joint.

As the clamped parts are raised in temperature, the edges begin to deform. At that moment, the torch is made to oscillate within 10 to 12 mm of the centre line of the joint at the rate of 1 or 2 oscillations per second, thereby heating the metal adjacent to the seam uniformly. When the metal at the joint is white-hot and a thin coating of molten metal appears on it, the parts are pressed together to obtain the desired upset. Then the burner is switched off, and the weld is allowed to cool in air. The structure of the weld metal is usually refined by normalising after the workpiece has cooled, when the metal is again heated by the same burner to 850-900° C and is allowed to cool in air.

(*e*) *Welding Defects.* When the edges of a joint are heated insufficiently or nonuniformly, the faces fail to fuse together across the entire section of the joint, and what is known as incomplete fusion, a very common defect of gas-pressure welding, occurs. Sometimes, it may result from insufficient upset pressure, rust, mill scale or dirt left at the joint. The latter factors are also responsible for weak welds.

17*

Poor alignment results in an offset joint. Excessive heating brings about grain growth in the near-weld area. Welds with overheated metal have low ductility and low impact strength. The effect of overheating may be removed by subsequent normalising.

A flame with an excess of oxygen is responsible for the oxidisation and burning of metal at the joint, when much porous scale is formed on the surface of the joint, and the strength of the weld is reduced appreciably.

(2) Open-butt welding. In gas-pressure welding by the open-butt method, the ends of the bars or rods to be welded may be cropped by oxygen cutting, so that an allowance of 15 to 20 mm is left for melting and upsetting. The kerf should be thoroughly cleaned with a chisel or emery cloth and wire brush to remove traces of slag and oxides. The bars are then brought together, a multiflame burner is fired, and the surface of the joint is heated. The distance between the burner tips and the metal surface on the upper side should be 1.5 or 2 mm smaller than on the underside, or the lower side of the joint will be heated more. While the joint is heated, the burner should be moved as much as the diameter of the bars being welded right and left of the centre line of the joint. The rate of motion should be 20 to 25 in the beginning and 50 to 60 per minute towards the end of the heating cycle.

When the metal surface is brought up to 1100-1200° C (an orange-yellow heat), the bars are withdrawn 15 or 20 mm, and the flame is made to play on the end faces of the bars. This also serves to remove slag from the joint, as it is carried away by molten metal. Then, without withdrawing the flame, the bars are pressed together. The upset pressure should be 3-3.5 kg/mm². After the bars have been given the desired upset to complete the weld, the upset pressure is removed. As soon as the weld metal has solidified, the bars may be taken out of the clamps. The ductility and impact strength of the weld metal are usually improved by heating the weld to 1150° C either in a hearth or with a burner, and the flash is quickly chipped out by a chisel, after which the weld is peened in a die under a hammer to the desired dimensions.

Torch soldering, hard-facing and flame hardening

TORCH SOLDERING

1. General

Soldering is a process of joining metals together in a solid state by means of alloys, called solders, which have a lower melting point than the metals to be joined.

In soldering, the surfaces to be joined together are not melted. Instead, they are raised to a point which is above the melting point of the solder. The molten solder wets the surfaces to be joined, spreads over them, penetrates into them, and solidifies, thereby forming the joint. Some of the solder is drawn between the grains of the parent metal mechanically. In addition, the joint incorporates a thin intermediate layer of solid solutions (included in the structure of the solder and parent metal) or their chemical compounds.

There are two general methods of soldering, namely, soft soldering and hard soldering also known as brazing. The former uses soft solders based on tin and lead, and the latter uses hard solders which are based on copper and silver.

Hard solders melt at above 550° C and have an ultimate tensile strength of up to 50 kg/mm². Soft solders melt at not over 400° C, and soft-soldered joints have an ultimate tensile strength of not more than 7 kg/mm².

Soft soldering is done either manually, by means of a heated copper bit or "iron" or a gas torch, or mechanically, by means of special units which both heat and solder the parts to be joined. The latter method has found wide use in large-scale production soldering.

When brazing on a limited scale, the parts to be joined are likewise heated locally and the hard solder is melted by means of air-gas or oxy-gas torches (torch brazing). In large-scale production brazing, wide use is made of techniques in which the parts to be joined are entirely heated in electric or gas-fired furnaces with a reducing atmosphere (furnace brazing), in molten baths (dip brazing), etc.

Brazing may be used for joining virtually all metals and dissimilar combinations of metals. The best results are obtained with cast iron, plain carbon and alloy steels, copper and its alloys, nickel and its alloys, and aluminium.

Soldering, both soft and hard, is widely used commercially, as the production rate with it is sufficiently high, the joints are tight, and the parent metal needs not be heated to a point where its structure is affected.

Among the demerits of the process are its applicability to a limited number of joint types, primarily, lap joints; the reduced strength and ductility of the joints compared with the parent metal; the dependence on scarce metals (tin, silver, etc.) for solder manufacture, especially when the joints are to be of high quality; and more thorough cleaning of the joints for satisfactory soldering. All this has limited the field of application for soldering in engineering industries.

2. Soldering Torches

Local heat in soldering may be supplied by either torches or blow-lamps. When the fuel is a mixture of acetylene and oxygen, use may be made of the ordinary ГС-53 welding torch.
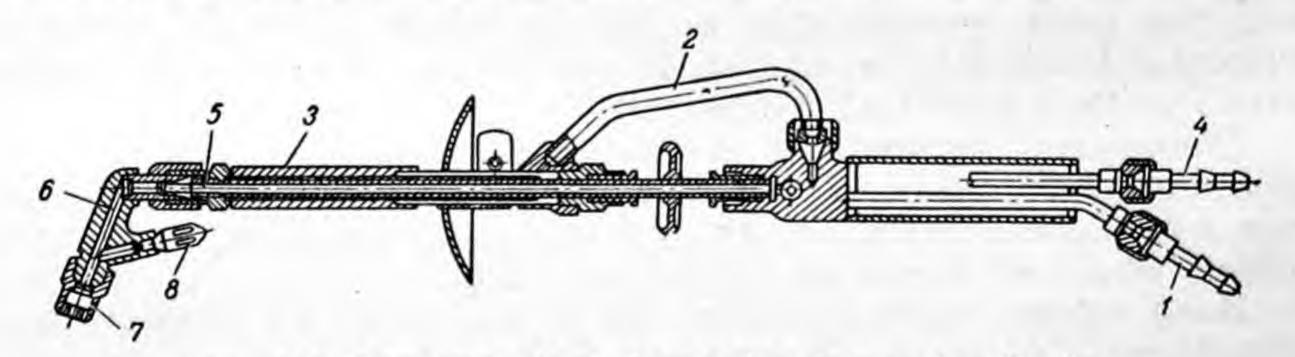
Fig. 116. Oxy-benz torch for preheating and soldering

In the latter case, the single-jet tip should be replaced by a mesh-type tip so that the heat may be spread over a larger area. The use of mesh-type tips increases the rate of soldering by as much as 10 or 15 per cent.

When instead of acetylene other fuel gases are used in a mixture with oxygen, such as town gas, propane, etc., conventional injector torches may be used with adequately changed ports for these gases.

Fig. 116 shows a soldering oxy-benz torch which operates on a mixture of oxygen and the vapours of petrol, kerosene or other liquid hydrocarbons. The performance of this type of torch is summarised in Table 29.

Table 29

Performance of Soldering Torch

Inlet pressure, kg/cm²		Consumption		
			petrol	kerosene
oxygen	fuel gas	oxygen, lit/hr	grams/ r	
3.0	1.5	750-800	480-510	520-550
4.0	2.0	750-950	480-560	520-600
5.0	2.5	750-1,100	480-700	520-780

Petrol or kerosene is pressure-fed from a tank of about 5 litres capacity via a nipple *1* to the torch tube *2* whence it enters the evaporator *3* packed with asbestos. Oxygen is fed via another nipple *4* into the central passage of the injector *5* to form a combustible mixture in the head *6*. As the mixture leaves the head by a mesh-type tip *7*, it burns in multitude of flames. Part of the mixture leaves by another tip *8* to heat the evaporator *3*.

3. Solders

For its strength a soldered joint depends above all on the properties of the solder used. Therefore, in choosing a solder, the following considerations must be taken into account.

(1) The melting point of a solder should be at least 60° C below that of the metals to be joined.

(2) A solder should be fluid when molten so as to be able to fill the clearance between the surfaces being joined and to wet them efficiently.

(3) A solder should not differ markedly from the metals to be joined in corrosion resistance.

Of the soft solders used commercially, the most common are tin-lead alloys, and of the hard solders, copper-zinc and silver alloys. Silver solders, apart from silver and copper, also contain zinc, phosphorus and cadmium. The composition and properties of soft solders are summarised in Table 30, and those of hard solders in Tables 31 and 32.

An addition of about 1 per cent tin to copper-zinc solders improves their wetting ability. Another positive effect of the tin addition is that it oxidises more readily and prevents the zinc from burning out. When added to copper-zinc solders, silicon acts in a similar way. Serving as a good deoxidiser in brazing,

Table 30

Soft Solders

Solder designation	Chemical composition, %, (remainder, Pb)		Impurities, * max.			Melting point, °C	Sp. gr.	Strength, kg/mm²		Application
	Sn	Sb	Cu	Bi	As			tensile	shear	
ПОС 90	89-90	0.1-0.15	0.08	0.1	0.05	222-183	7.57	4.3	2.70	Inside seams on food-processing and medical equipment
ПОС 40	39-40	1.5-2.0	0.1	0.1	0.05	235-183	9.31	3.2	3.67	Brass, iron and copper-wire soldering
ПОС 30	29-30	1.5-2.0	0.15	0.1	0.05	256-183	9.69	3.3	2.90	Brass, copper, iron, zinc and zinc-plated sheet, tinplate soldering; manufacture of instruments, radiators, equipment
ПОС 18	17-18	2.0-2.5	0.15	0.1	0.05	277-183	10.23	2.8	2.52	Lead, iron, brass, zinc-plated iron soldering; manufacture of automotive parts, utensils and household goods
ПОСС 4-6 (antimonium-bearing)	3-4	5-6	0.15	0.1	0.05	265-245	10.7	5.9	—	Tinplate, brass, iron, copper, and lead soldering over riveted joints; repair of car bodies, etc.
ПОС 50	49-51	—	—	—	—	209-183	8.87	3.6	3.54	Aircraft radiator soldering
ПОС 25 **	24-25	—	—	—	—	265-183	9.94	2.8	2.85	Same as ПОС 30

Notes: * The impurity content should not exceed 0.02 per cent of iron and sulphur (each), 0.002 per cent (traces) of aluminium and zinc.
** Nonstandard solder.

Table 31

Silver Solders

Solder designation	Chemical composition, %					Melting point, °C	Sp. gr. when cast	Strength of soldered joints, kg/mm²		Application
	Ag	Cu	Cd	P	Zn			shear	rupture	
ПСр 10	10	53	—	—	remainder	830	8.5	—	—	Brazing of parts subjected to subsequent heating up to 800° C
ПСр 12	12	36	—	—	Ditto	785	8.5	—	—	Brazing of copper and brass containing over 58 per cent copper
ПСр 25	25	40	—	—	Ditto	765	8.9	25	23	Brazing of small parts where clear joints and high ductility are essential
ПСр 45	45	30	—	—	Ditto	720	9.3	28	32	Brazing of critical copper and bronze parts where joints of high electrical conductivity are essential
ПСр 65	65	20	—	—	Ditto	740	9.6	—	—	Brazing of band saws
ПСр 70	70	26	—	—	Ditto	780	9.8	—	—	Brazing of current-carrying parts
Phosphorus-bearing	15	80	—	5	Ditto	645	8.45	27	27	Brazing of copper and copper alloys. Self-fluxing grade
Cadmium-bearing	50	15.5	18	—	16.5	595	—	—	45	Brazing of practically all metals. High strength; low melting point

Table 32

Copper-tin Solders

Solder	Chemical composition, %				Sp. gr.	Melting point, °C	U. T. S., kg/mm²	Elonga-tion, %	Application
	Cu	P	Fe	Zn					
ПМЦ 36 (copper-zinc)	36±2	0.5	0.1	Rema- inder	7.7	825	—	—	Brazing of brasses Л59 and Л59-1, special brasses containing 59 per cent or less copper
ПМЦ 48 (copper-zinc)	48±2	0.5	0.1	Ditto	8.2	870	21	3	Brazing of brass Л62
ПМЦ 54 (copper-zinc)	54±2	0.5	0.1	Ditto	8.3	876	26	4	Brazing of copper, tom- bac, bronze and steel
Brass Л 62	60±2	0.1	0.2	Ditto	8.6	905	35	30	Ditto

silicon readily oxidises to silica, the silica floats to the surface of the molten metal and protects the zinc of the solder from oxidation. Some 0.12 per cent silicon is usually added to copper-zinc solders. An addition of phosphorus to copper-zinc solders makes them more fluid and acts as a deoxidiser in soldering.

Phosphorus-copper solders are employed in electrical engineering as substitutes for silver alloys for brazing copper, brass and bronze. Their advantage is that they require no flux. On the other hand, phosphorus-copper solders are unsuitable for brazing ferrous metals because of the low wetting ability and the formation of a brittle zone at the interface between the joint and the parent metal. Phosphorus-copper solders make good joints with a strength comparable with that of silver alloy joints. Because of embrittlement, however, their use is limited to cases where shock loads are nonexistent. The solders in this group are extremely fluid. The most commonly used of the phosphorus-copper solders contains 7 or 8 per cent phosphorus and the remainder is copper. Its ultimate tensile strength is 17 kg/mm² and its shear strength is 28 kg/mm².

For the most part, solders are obtainable in rod, wire, ribbon, sheet or foil form. The only exception is the copper-zinc group of solders which, because of increased brittleness, comes as powders of different grain sizes.

Special solders are employed to solder or braze aluminium. One of the soft solders used for the purpose contains 50 per cent zinc, 45 per cent tin, 5 per cent aluminium. Another contains 20

to 25 per cent zinc, 2 to 6 per cent aluminium, and the remainder tin. Unfortunately, soldered joints on aluminium made with soft solders have poor corrosion resistance.

Aluminium and its alloys are brazed with solders containing not over 70 per cent aluminium. These solders are ternary alloys of the Si-Cu-Al type with a melting point of 525° C. Their composition is 5.2-6.5 per cent Si; 26-29 per cent Cu, and the remainder aluminium. A special alloy, designated 34A, has been developed by S. N. Lotsmanov. It is made by smelting at 650-700° C an equal amounts of No. 1 alloy (50-60 per cent copper, the remainder aluminium) and of No. 2 alloy (8-14 per cent silicon, the remainder aluminium). The resultant solder contains 6 per cent silicon, 28 per cent copper, and the remainder aluminium. It melts at 525° C.

Brazed joints in aluminium and its alloys have satisfactory resistance to corrosion.

4. Fluxes

Soldering or brazing fluxes protect the molten solder and the surfaces to be joined from oxidation, dissolve the oxides that form by heating, and cause the molten solder to spread by breaking down the surface tension of the molten alloy. A successful flux should meet the following requirements:

(1) Its melting point should be below that of the solder and its vaporisation point should be above the melting point of the solder.

(2) It should actively dissolve the oxides forming on the surface of the metal being joined and the solder.

(3) It should not react chemically with the parent metal and the solder.

The compositions of the most commonly used fluxes are given in Tables 33 and 34.

Most often, fluxes are used in powder form. When a flux is applied to a heated surface, it softens and sticks to the metal surface. Though on a smaller scale, use is also made of vapour fluxes. Vapour fluxes make it possible to use fluxes and solders more economically, with the flux covering the surfaces to be joined more uniformly. Vapour fluxes are usually borates of hydrocarbons of low vaporisation point. In the process of soldering, the stream of acetylene or the other fuel gas used is allowed to bubble through the dissolved flux and to pick up its vapours which are burned in the flame to form boron oxide. Therefore, fluxing elements appear directly in the flame.

Table 33

Fluxes for Soft Soldering

Composition	Application
10-30 per cent zinc chloride, 90-70 per cent water	Soldering of iron, steel, copper, copper alloys
25-30 per cent zinc chloride, 5-20 per cent sal-ammoniac, 50-70 per cent water	Same metals, but makes a more efficient flux
20 per cent zinc chloride, 5 per cent sal-ammoniac, 74 per cent petroleum jelly, 1 per cent water	Soldering paste for same metals
Saturated solution of zinc chloride in hydrochloric acid	Soldering of stainless steel
30 grams powdered resin dissolved in 100 cm³ orthophosphoric acid (1.6—1.7 sp. gr.) and 400 cm³ alcohol or ethyleneglycol *	Soldering of stainless steel with pure tin or tin-lead solders at 290—300° C
Resin	Soldering of copper and its alloys
Saturated aqueous solution of 34 per cent zinc chloride, 33 per cent methanole and 33 per cent glycerine	Soldering paste for use with blow-lamps

Note * Developed by S. N. Lotsmanov and A. S. Medvedev

A special group of soft solders for aluminium soldering has been developed by S. N. Lotsmanov:

No. 1	Zinc chloride	85 per cent
	Sal-ammoniac	10 per cent
	Sodium fluoride	5 per cent
No. 2	Zinc chloride	90 per cent
	Sal-ammoniac	10 per cent
No. 3	Zinc chloride	95 per cent
	Sodium fluoride	5 per cent
No. 4	Zinc chloride	90 per cent
	Sal-ammoniac	8 per cent
	Sodium fluoride	2 per cent

The resultant surface coating protects the aluminium from oxidation, while the aluminium chloride is sublimated into a

Table 34

Fluxes for Hard Soldering

Composition	Application	Notes
Crystallic or fused borax, 100 per cent	Brazing of copper, brass, bronze, steel	—
72 per cent fused borax, 14 per cent common salt, 14 per cent calcined potash	Ditto	Very fluid; also used in silver alloy brazing
80-90 per cent fused borax, 20-10 per cent fluorspar or common salt or zinc chloride or potassium fluoride	Ditto	Used in silver alloy brazing
90 per cent fused borax, 10 per cent boric acid	Ditto	Has increased melting point
50 per cent fused borax, 50 per cent boric acid	Brazing of stainless steel	Soldering paste when used with zinc chloride solution
90-93 per cent fused borax, 7-10 per cent powdered metallic magnesium	Ditto	Magnesium is added to molten borax; flux is ground after cooling
70 per cent boric acid, 21 per cent borax, 9 per cent calcium fluoride	Brazing of stainless steel and heat-resistant alloys with high-melting solders which melt at 850-1,100° C	Used in powdered form or as a paste mixed with water

Table 34 (continued from page 271)

Composition	Application	Notes
35 per cent boric anhydride, 42 per cent dehydrated potassium fluoride, 23 per cent potassium fluor-borate	Brazing of structural and stainless steels, heat-resistant and copper alloys with silver solders	—
60 per cent borax, 38 per cent zinc chloride, 2 per cent potassium permanganate	Brazing of cast iron	—

white smoke. Small aluminium articles are dipped through a layer of flux into the molten solder at 420-450° C. In such a case, the flux to use should be No. 3. Aluminium brazing requires special fluxes which are similar to those used in aluminium welding where they form low-melting fluorides and chlorides which readily dissolve the surface aluminium oxide.

Fluxes for aluminium brazing should readily dissolve the surface oxide (Al_2O_3) at 450-550° C, which is lower than in aluminium welding. This is accomplished by adding some lithium chloride to the flux. Satisfactory results have been obtained with the 34A flux, developed by S. N. Lotsmanov, which contains 29-35 per cent lithium chloride, 6-10 per cent zinc chloride, 9-11 per cent sodium fluoride, the remainder potassium chloride. This flux melts at 420° C. The potassium chloride may be replaced by sylvite or sylvinite; the sodium fluoride by potassium fluoride or lithium fluoride; the zinc chloride by lead chloride; and the lithium chloride partly by magnesium chloride.

Cracks in cast aluminium alloys are soldered with a solder containing no lithium, 8-10 per cent sodium fluoride, 10-15 per cent barium chloride, 15-20 per cent sodium chloride, 30-40 per cent zinc chloride and the remainder potassium chloride.

5. Soldering Procedure

(1) Types of Soldered Joints. In soft soldering, the only type of joint used is the lap joint. The lap should be 3 to 60 mm, depending on the thickness of the metal and the working conditions of the joint. On thicknesses from 2 to 5 mm and a pressure

of up to 5 atm (gauge), the lap should be at least 40 mm and the joint should be designed to work in shear. In brazing, the most commonly used type of joint is also the lap joint. The other types of joints used in soldering practice are shown in Fig. 117.

The factors governing the strength of a joint are the area of the joint (or lap), the fit-up of the members to be joined and the clearance between them. The clearance, though, is of importance
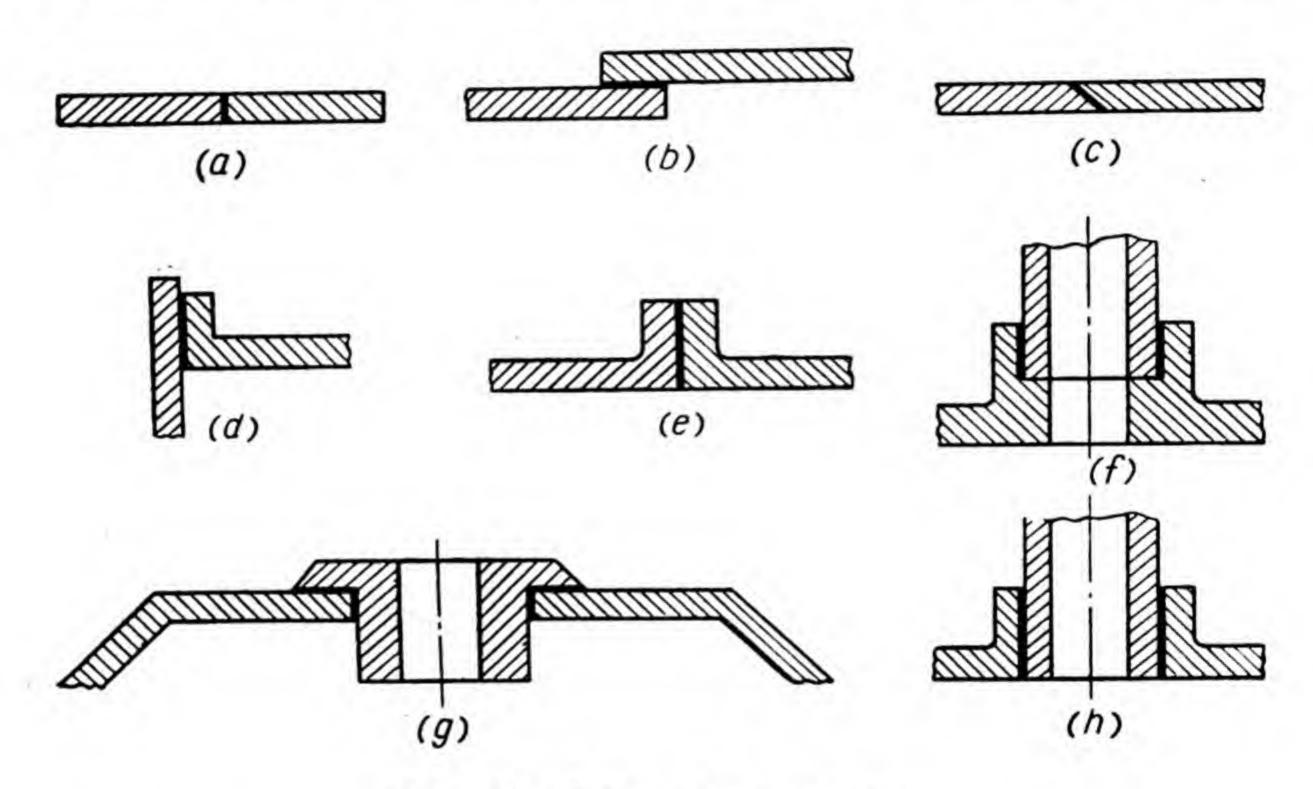

Fig. 117. Soldered joint designs:
a — butt; *b* — lap; *c* — scarf; *d* — corner; *e* — flange; *f, g, h* — sleeve

to the strength of soldered joints in high-strength metals, while in low-strength metal it is of minor consequence.

As a rule, the clearance should be kept to a minimum to facilitate capillary attraction of the molten solder and to produce the strongest joint. When the clearance is sufficiently small, the metal of the joint is an alloy of the parent metal and the solder formed by mutual diffusion. For tubular members, the clearance should be 0.2-0.25 mm. In soldering with silver alloys, it should be 0.05-0.03 mm.

(2) Edge Cleaning. The surfaces to be joined should be cleaned of dirt, mill scale, oxides and grease, as any film prevents the wetting of the metal with the solder. Cleaning may be done either

mechanically or chemically. Chemically, oxides are removed by pickling the surface with an acid, which is followed by thorough washing and drying. Mechanically, this is done with brushes, fine-grained emery cloth, etc. Sand-blasting should be avoided, as the sand markings left on the metal surface prevent the wetting of the joint surfaces.

(3) Soldering Technique. In soft soldering, first some flux is applied to the cleaned surface and the joint heated with the torch or blow-lamp. As the flux vaporises, the solder melts and flows into the joint clearance; the flame is withdrawn, and the solder is allowed to solidify. A thin layer of solder can be flushed over the joint surface of one or both members before assembling to facilitate the flow of the solder into the clearance.

In brazing, after the joint surfaces have been properly prepared, the members to be joined should be clamped in a jig or a fixture giving easy access to the joint from all sides. The clearance should be set in advance, as should the overlap in the case of lap joints. Heating should be done with the broad part of the flame. In the case of an oxidising flame, a denser joint is usually obtained than with a reducing flame, but its appearance is not so good as in the latter case. The operator, however, will find it easier to work with a flame having a slight excess of acetylene, as there is no danger for him to overheat the solder, and the flame, being soft, will not blow off the flux. When joining members differing in thickness or thermal conductivity, the flame should be made to play on the member that is thicker or more conductive thermally.

The flame should be played on the joint, as the molten solder tends to flow where the metal is hotter. Some flux should be applied to the heated joint surfaces just before brazing. Some flux should also be applied to the solder which should be slightly heated. The solder should be melted by the heated members as the solder rod is made to touch the edge of the member, and never by the flame. After a joint is completed, the solder and metal should be allowed to cool slowly. Nonferrous metals may be cooled in water upon brazing. If the flux includes borax, the quick cooling will facilitate its removal from the surface of the joint. Fluxes consisting of feldspar can be removed by washing the brazed part in hot water. Fluxes incorporating borax are removed by pickling in a 10-per cent solution of sulphuric acid and by washing with water.

Aluminium brazing involves the cleaning of the joint surfaces with a fine-wire brush; flushing with petrol or a 10-percent

solution of NaOH and pickling with a solution of nitric acid; heating of the joint with a torch to the melting point of the flux; the application of some flux to the joint; the placement and melting of the solder in the joint; the cleaning of the joint to remove surplus flux and solder. Surplus flux is removed by washing the joint with warm water and by wiping it dry with a piece of clean cloth. Aluminium brazing uses only lap joints, and the joints thus obtained lack in ductility.

CHAPTER XVII

TORCH HARD-FACING

1. Hard-facing Materials

Hard-facing, or surfacing, is a method of applying an alloy material to a metal part so as to form a protective surface to resist abrasion, corrosion, heat, impact or any combination of these factors. The parts that are usually hard-faced are drilling tools, cutting tools, excavator dipper teeth, hydraulic dredger cutters, etc. The hard-facing alloy may be applied by arc and oxy-gas welding. The hard-facing deposit usually has a hardness of 40 to 95 Rockwell C.

For the most part, hard-facing materials cannot be heat-treated. As a rule, they possess low ductility and appreciable brittleness, for which reason they cannot be forged. Instead, they are finished by grinding. In hard-facing, they develop cracks, as they retain reduced ductility at elevated temperatures.

For the purpose of identification, it is customary in the U.S.S.R. to classify the hard-facing materials used commercially into cast (such as Sormites); powdered materials (Stalinite and Wocar); sintered (Pobedite); and fused (tungsten carbides).

Cast materials are alloys of cobalt, chromium and tungsten carbides, and sometimes small quantities of iron. They are rather expensive because of the scarce tungsten and cobalt. This has spurred the development of substitutes (known in the U.S.S.R. under the trade name of Sormites) containing neither tungsten nor cobalt. These substitutes are alloys of iron and chromium with a nickel content of up to 5 per cent. The necessary hardness is attained by an addition of chromium carbides. Chromium runs 25-31 per cent in Sormite No. 1, and 13.5-17.5 per cent in Sormite No. 2. The latter is less brittle and can, when annealed, be machined and quenched. The Sormite deposit has a hardness of 40-50 Rockwell C.

Wocar, a hard-facing material in the powdered group, is a mixture of ferrotungsten and carbon, specially treated. Wocar is very expensive and gives a hardness of 60-63 Rockwell C.

Stalinite does not contain any tungsten for which reason it is cheap to make. It is composed of carbon, ferrochromium and ferromanganese. The hardness attainable with Stalinite is 56-57 Rockwell C. Owing to its low cost, lower melting point (1300-1350° C) and high hardness, Stalinite has found a broad field of commercial application.

Sintered materials, or, cermets are obtainable as blanks, bits, tips and inserts and have a Rockwell C hardness of 86 to 91. They are copper- or brass-brazed to the shanks of cutting tools to form tool points. Cermets are basically alloys of tungsten and titanium carbides with cobalt and nickel as binder materials.

Fused carbides (practically, pure tungsten carbides) are the most hardest of all (92-94 Rockwell C) and have a melting point of about 3000° C. They are obtainable as bits with sharp edges which are welded into the shanks of tools with the space between the bit and the shank being filled with cast or powdered alloy. In service, the sandwiched filler wears out quicker than the insert that forms the cutting edge. These alloys mainly go to make drilling tools for hard rock.

Hard-facing also uses rods which consist of a low-carbon steel shell or casing with an outside diameter of 6 mm and a wall thickness of 0.5 mm in which are enclosed tungsten carbide grains. These rods are mainly used for hard-facing with the oxy-acetylene flame. When properly deposited, the hard-face contains about 85 per cent tungsten carbide and 15 per cent iron and has a hardness of about 85 Rockwell C. In the Soviet Union they are designated T3 and are used to hard-face oilfield drilling tools.

2. Hard-facing Procedure

The oxy-acetylene flame can be used for hard-facing with cast alloys and T3 rods and for brass-brazing cermet bits to steel shanks. Powdered hard-facing alloys are difficult to deposit with the oxy-gas flame, as they are blown off by the flame in the course of surfacing.

The hard-facing procedure with cast alloys and T3 rods is as follows.

The part to be hard-faced should be examined for cracks or other defects, as they weaken its strength under the hard deposit. If there are some defects, they should be welded up prior to surfacing. Hardened parts should first be annealed at 750-900° C to relieve locked-up stresses.

The hard-facing deposit is very brittle. Therefore, where impact loads are likely to be encountered, the deposit should be not more
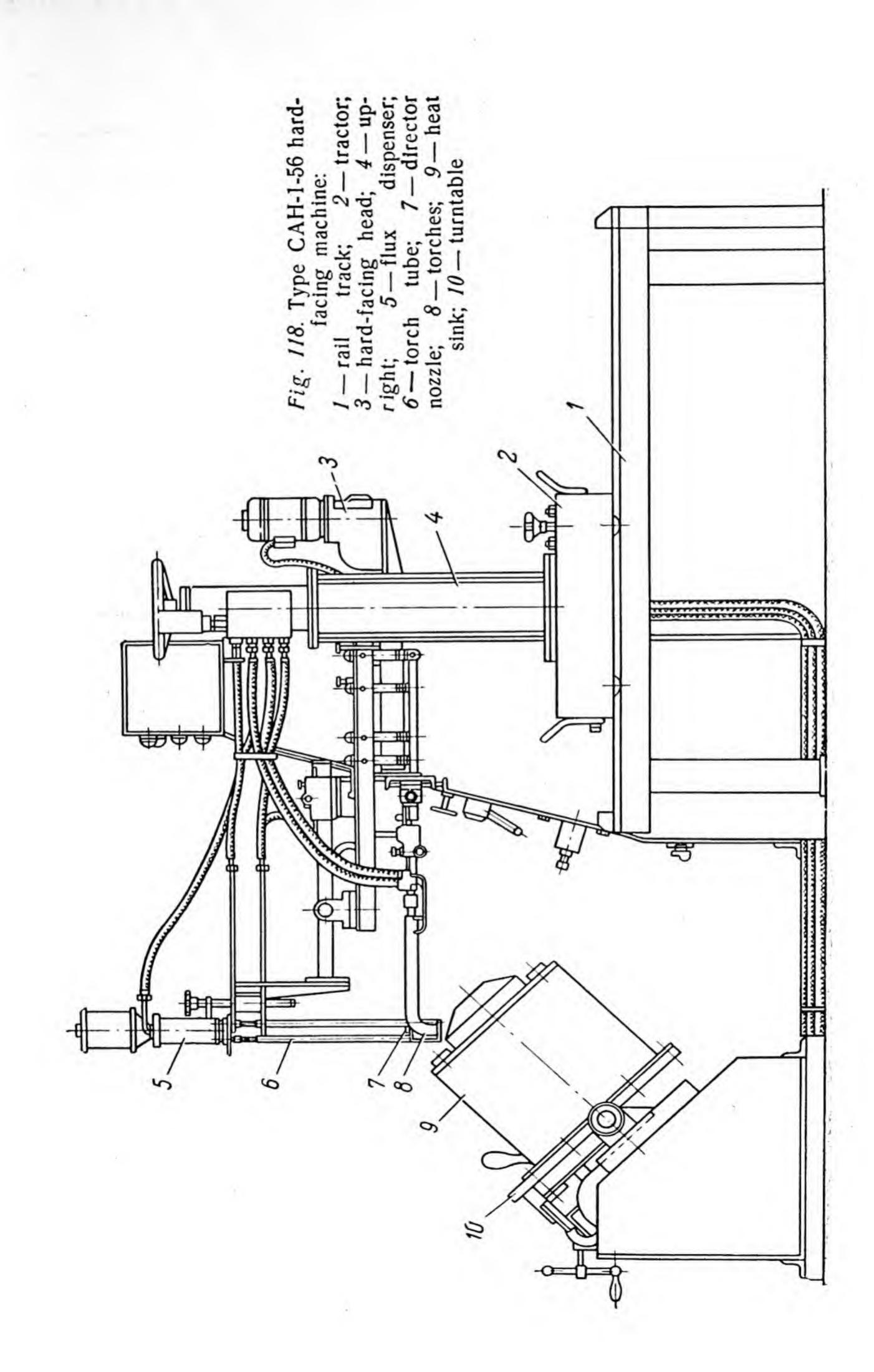
than 2.5 or 3 mm thick. Where parts are to stand attrition, the hard-facing deposit should be not more than 4 to 8 mm. In building up heavily worn parts, it is good practice to deposit a layer of low-carbon steel which is then hard-faced with a suitable alloy. In the case of parts from special alloy steels (such as high-manganese steels), a better bond is obtained by first depositing a layer of low-carbon steel which is then followed by a layer of a hard-facing alloy. When necessary, a recess is machined or a chamfer made on the part to be hard-faced. The chamfer should be the same depth as the metal deposit. The sharp corners of the chamfer should be rounded off. The chamfer should be 5 to 10 mm wide.

When hard-facing with an oxy-gas flame, the part should be preheated to 500-750° C. This protects the part from warping in the course of surfacing and improves the quality of the latter. Hard-facing should be carried out in the downhand position, using a flame which has an excess of acetylene. The excess of acetylene serves to carburise the metal surface, thereby reducing its melting point. When a thin film of molten metal appears on the surface (the metal is said to sweat out), the end of the hard-facing rod is introduced into the flame. In melting, the hard-facing material spreads over the steel to form a strong overlay of hard alloy. With this technique, the parent metal is melted to a depth of not more than 0.3 or 0.5 mm. This reduces the dilution of the deposit by the parent metal, and the hard-face retains the mechanical properties of the source material. For the same reason, when using T3 rods, it is essential to limit the depth to which the parent metal is sweated.

Stellite-type or Sormite hard-facing rods usually have a diameter of 5 to 7 mm. In hard-facing carbon steels (say, of the tool grade) or cast iron, borax may be used as flux. After the hard-facing operation the workpiece should be allowed to cool slowly in sand. The low rate of cooling is preferable in all cases, as this prevents cracking in the hard-face. After cooling, the hard-faced part is given the desired shape and dimensions either by machining (if the hard-face can stand temper and quenching) or by abrasive grinding wheels.

Cermet inserts cannot stand heavy overheating. When they are brazed to tool shanks by the oxy-gas flame, it is good practice to preheat both the insert and the shank to a red heat. The solder may be brass wire, and the flux may be either borax or a mixture of borax and boric acid.

Hard-facing operations have now been mechanised and automated by the use of the CAH-1-56 hard-facing machine developed by the Autogenous Welding Research Institute (Fig. 118). The

Fig. 118. Type CAH-1-56 hard-facing machine: 1 — rail track; 2 — tractor; 3 — hard-facing head; 4 — upright; 5 — flux dispenser; 6 — torch tube; 7 — director nozzle; 8 — torches; 9 — heat sink; 10 — turntable

machine which is operated from a control panel, can hard-face bodies of revolution (valves, bushings, shafts, etc.). The hard-facing head of the machine has two torches, a flux dispenser, and a director nozzle to feed a hard-facing rod. The part to be hard-faced is placed in the heat sink of the turntable. Both the rod and the torches can oscillate within 1.5 to 12 mm. The flux for the BK3 alloy which is of the Stellite class should consist of 25 per cent dehydrated borax, 25 per cent boric acid, and 50 per cent fluorspar. The powdered mixture should pass through a sieve having 1,600 meshes per square centimetre and should have a moisture content of not over 5 per cent.

Technical Data of CAH-1-56 Hard-facing Machine

Diameter of workpiece	5-250 mm
Width of hard-face	up to 40 mm
Thickness of hard-face	up to 5 mm
Diameter of hard-facing rod	5-8 mm
Speed of hard-facing	25-100 mm/min

Flow rate:

acetylene	375-1,800 lit/hr
oxygen	375-1,800 lit/hr
compressed air (nitrogen or carbon dioxide)	12-60 lit/hr
cooling water	1.5-5.0 lit/hr

Overall dimensions:

length	2,650 mm
width	2,600 mm
height	1,750 mm

FLAME HARDENING

1. General

Flame hardening is a process of heating the surface of a ferrous material to above the upper critical point, Ac_3, with an oxy-acetylene torch and subsequently giving a very fast quench to produce a hard martensitic surface. The core of the material retains its original structure and all the physical and mechanical properties. The depth of hardening varies with the chemical analysis of the steel, the duration and intensity of heating, and the rate of cooling at the surface.

As some of the heat runs away from the surface into the bulk of the metal, the flame-hardened material ineluctably undergoes natural temper which produces some intermediate structures. Fig. 119 relates the hardness of the flame-hardened surface and
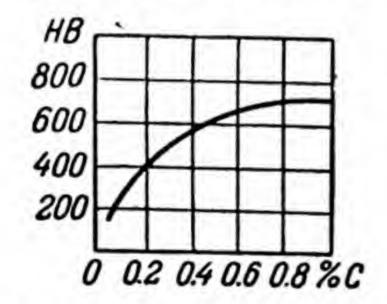
Fig. 119. The effect of carbon content on the hardness of the flame-hardened layer in steel
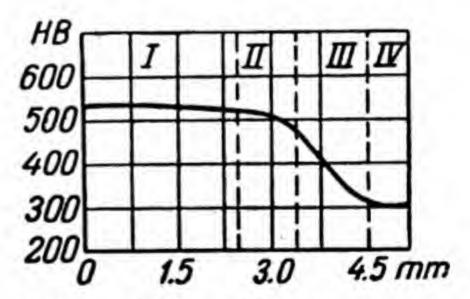
Fig. 120. Changes in hardness and structure with the depth of the flame-hardened layer in steel 40:
I — martensite (hardened layer); II — troosto-martensite (fusion zone); III — troostite (fusion zone); IV — pearlite and ferrite (core)

the carbon content of the original material. Fig. 120 plots the hardness and structure of the material against the depth of the surface treated. If the surface is to have sufficient hardness, the original steel should have 0.4 to 0.55 per cent carbon.

The flame-hardening process may advantageously be applied to crankshaft journals, valve-gear cams, piston pins, railroad crossings, gear teeth, stone-cutting tools, the guides of metal-cutting machines, etc. As compared with full hardening, the flame-hardening process is almost entirely free from distortion both on heating and cooling.

In addition to an oxy-acetylene torch, usually of the multiflame type and fitting the contour of the part to be treated, flame hardening can be accomplished by means of induction heating, resistance heating and electrolyte heating.
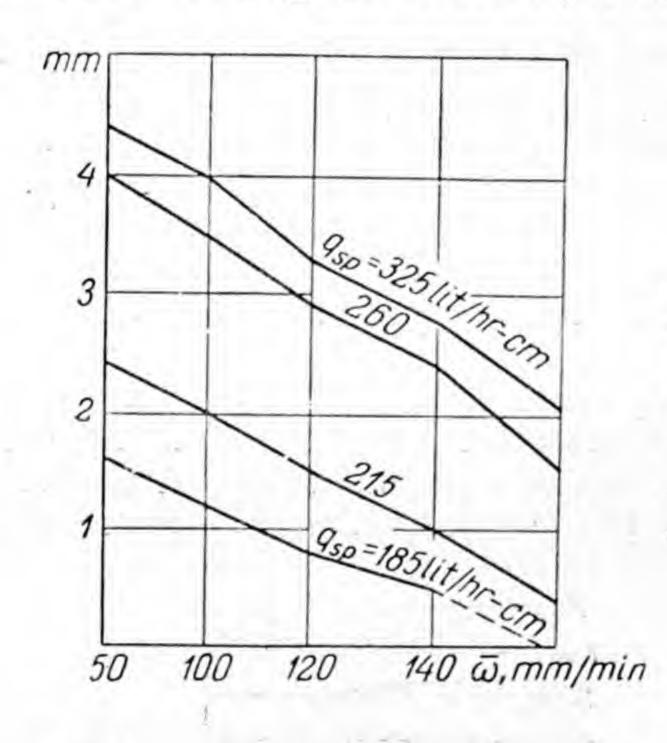

Fig. 121. The depth of the hardened layer as a function of specific heat input per unit length, q_{sp}, and rate of torch travel, ω, in flame hardening by the flat progressive method

There exist four basic methods of flame hardening, namely (1) the stationary method (both the torch and tho work are stationary); (2) the flat progressive method (in which either the work or the torch may move); (3) the spinning method (the torch is stationary and the work is rotated) and (4) the progressive spinning method (the flame or flames travel in a direction parallel to the axis of revolution). Methods (1) and (3) are best suited for small parts, while methods (2) and (4) come in handy where the work is long. The stationary method normally uses multirow torches in which the flame plays either on a limited area or the entire surface of the part being treated. The flat progressive and spinning methods employ in-line torches mostly having one or two rows of tips. The progressive spinning method involves the use of single-row in-line or ring torches.

The main advantage of flame hardening with an oxy-acetylene torch is the simplicity of the equipment and the possibility of using the process under any working conditions. Apart from ace-

tylene, the fuel gas may be city gas, coke-oven gas, natural gas, refinery gases, propane, petrol and kerosene vapours.

The depth of the hardened surface varies with the speed of torch travel with respect to the work and specific heat input, expressed in terms of acetylene rate of flow, in lit/hr per centimetre of metal width. Fig. 121 presents these data for a steel-45 bar 58 mm in diameter. Referring to the curves, the depth of the hardened surface decreases as specific heat input is reduced and the speed of torch travel is increased.

2. Flame-hardening Equipment and Procedure

Fig. 122 shows a Type МЗГ torch employed to flame-harden flat surfaces. This torch comes complete with assorted multi-orifice interchangeable tips. The performance data of the various tips are summarised in Table 35.
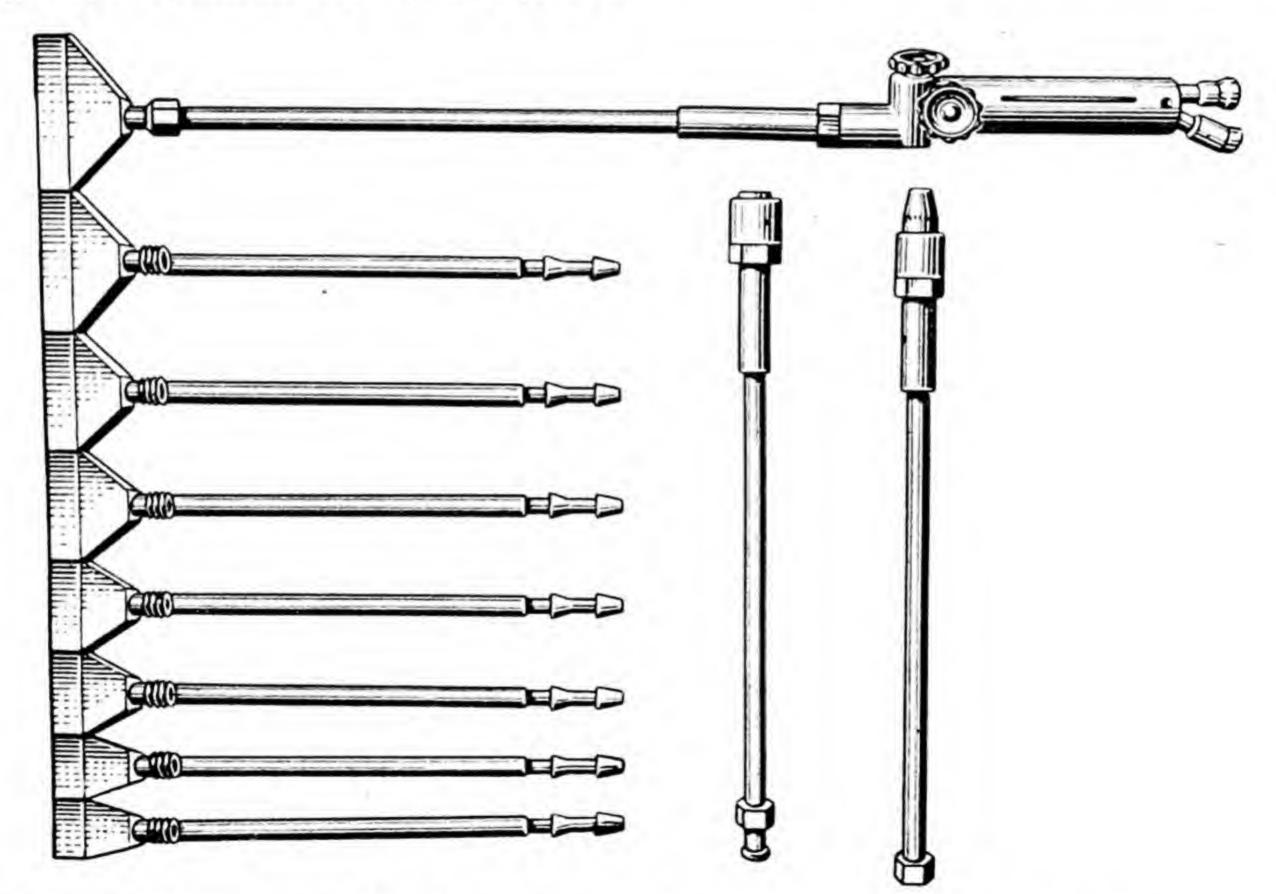

Fig. 122. Type МЗГ torch complete with interchangeable tips for flat and spinning flame hardening

All of the tips use oxygen at 2 to 4 kg/cm², and acetylene at 0.02 to 0.7 kg/cm². The rate of acetylene flow is 300–350 lit/hr per centimetre. The maximum length of the head is 675 mm, while its weight ranges from 1.16 to 1.3 kg, depending on the tip used. The

Table 35

Performance Data of Flame-hardening Tips

Tip width, mm	Hardened width, mm	Acetylene flow, lit/hr	Oxygen flow, lit/hr	Tip width, mm	Hardened width, mm	Acetylene flow, lit/hr	Oxygen flow, lit/hr
30	25	880	1,100	60	60	2,000	2,500
35	30	900	1,200	65	65	2,300	2,800
45	40	1,300	1,700	75	75	2,400	2,900
55	50	1,600	1,900	85	85	2,750	3,100

torch also has quench holes which spray water on both the work and the nozzles. The production speed with this torch is 1,500 to 8,000 cm²/hr with the surface hardened to a depth of up to 1.5 mm and the rate of head travel of 80 to 150 mm/min.
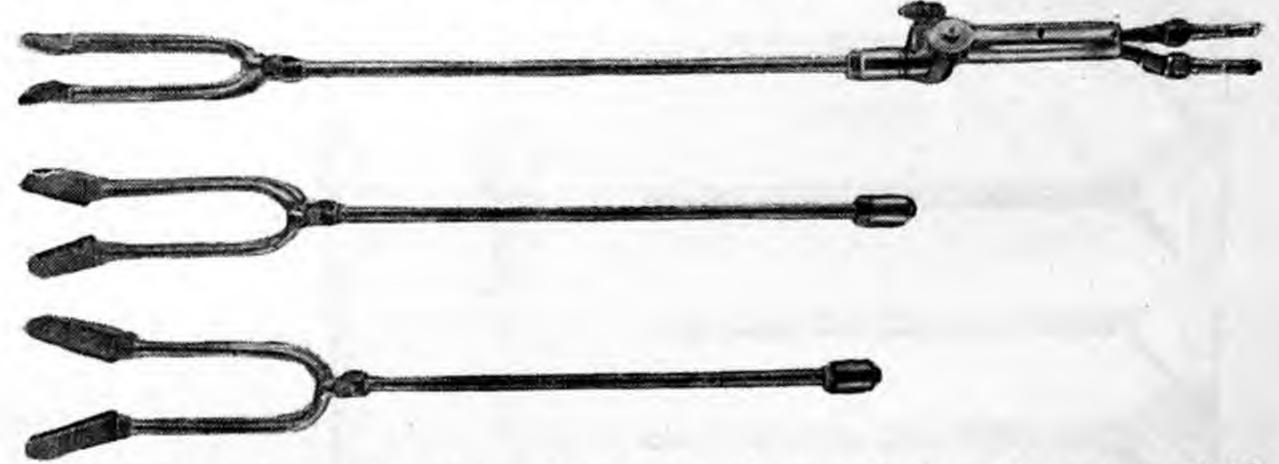
Fig. 123. Totches for the flame hardening of gear teeth with moduli of 10, 20 and 30

The quench in flame hardening torches can be provided either by jets of water, or air or quenching fluid. The quench device is usually a tube or a passage with a number of holes. The quench device may be made either integral with or external to the flame-hardening torch.

Fig. 123 shows a torch for flame hardening gears with moduli of 10, 20 and 30. The rate of acetylene flow is 1,300 to 1,350 litres per hour. The flame output is maintained at 350 lit/hr-cm, with the torch travelling at 180 to 100 mm/min (hardening speed). The rate of water flow for quenching is 0.65 to 0.9 lit/min. This type of torch is employed for the mechanised flame hardening of gears in a head developed by the Autogenous Welding Research Institute (Fig. 124). The head consists of a frame *12* which houses the

vertical motion of the upright *10*, actuated by an electric motor *11*. The upright carries a torch arm *4*, a flame-hardening torch *6*, lighter *7*, and a pneumatic drive *8* to move the torch in a horizontal plane. When the motor *11* actuates the upright *10*, the torch
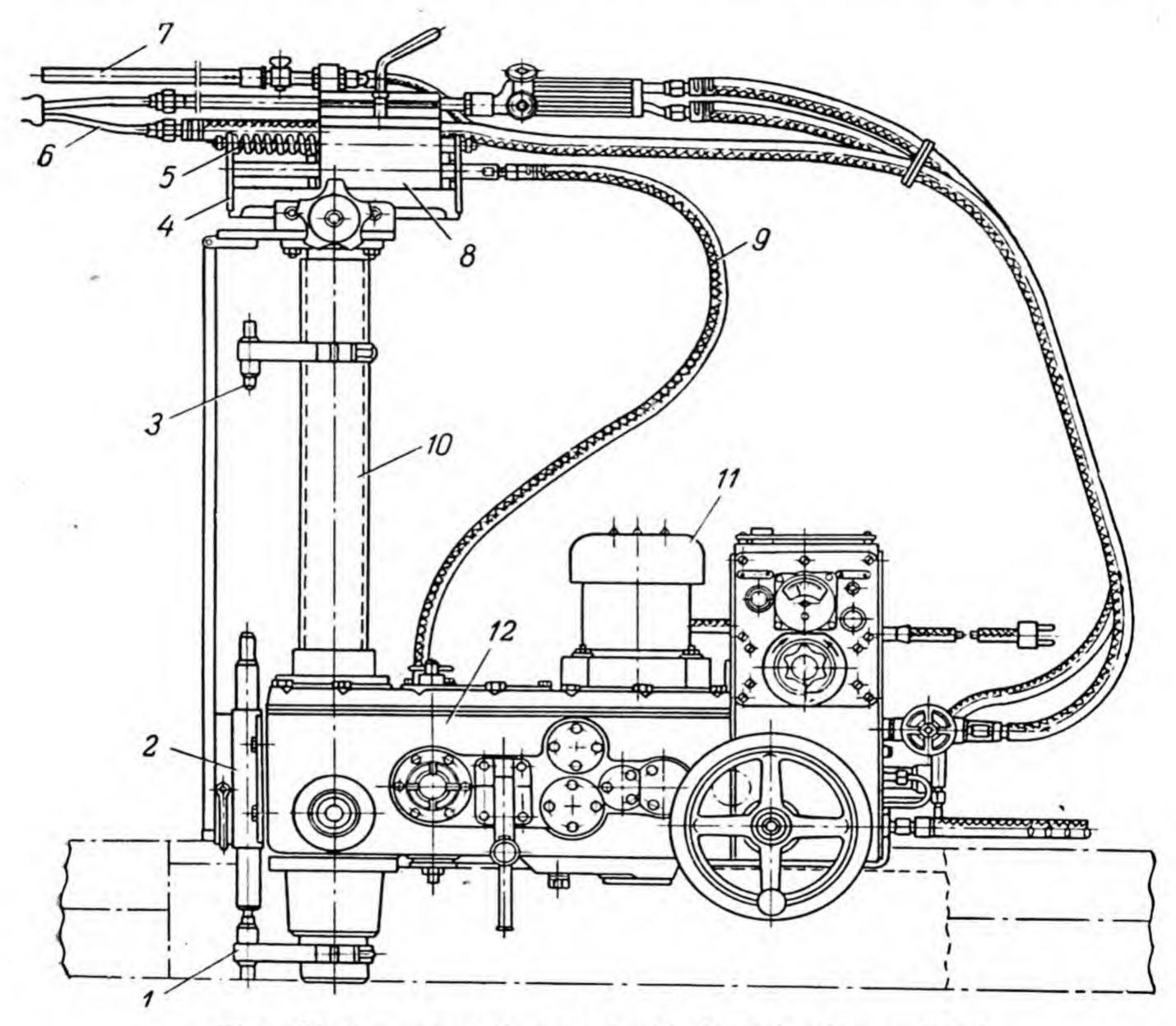
Fig. 124. An automatic gear-tooth flame-hardening head

is on an upward working stroke. In this case, a hose *9* supplies air to the cylinder of the pneumatic drive *8* to hold the torch near the surface of the gear tooth being hardened. When the upright *10* reaches its uppermost position, the limit stop *1* actuates the spindle of an air valve *2* which shuts the air supply off the pneumatic switch and the latter stops the flow of gas to the torch and of air to the cylinder of the drive *8* in that order. At the same time a spring *5* withdraws the torch from the gear tooth, the pneumatic

coupling of the drive disengages, and the upright *10* goes down. When the upper limit stop *3* presses the top end of the spindle in the valve *2*, the upright *10* is again reversed to go up, and the gas switch operates to admit gas to the torch. The lighter *7* sets fire to the gas mixture, the air pressure brings the torch within a short distance of the next tooth, and the upright *10* rises, flame hardening the tooth on the way. The head also incorporates a device which moves the gear one tooth on each time the upright is reversed.

Cylindrical surfaces are flame-hardened by in-line multiple torches like the ЛГ-200, ЛГ-400 and ЛГ-600. They are of the composite type and can be easily made broader by means of standard lengths of tips and by using an appropriate injector. The orifices in the tips are arranged in two rows, the spacing between the orifices being 5 mm and between the rows 4 mm. Each orifice is 0.8 mm in diameter. The tips are independently cooled by water. The performance data of these torches are given in Table 36.

Table 36

Performance Data of In-line Multiflame Torches

Characteristic	ЛГ-200	ЛГ-400	ЛГ-600
Hardened width, mm .	200	400	600
Number of interchangeable tips	5	8	8
Hardened width settings, mm	100, 125, 150, 175, 200	250, 300, 350, 400	450, 500, 550, 600
Pressure, kg/cm²:			
oxygen	2-5	2.5-5	3.5-5
acetylene	0.1-0.7	0.1-0.7	0.1-0.7
Flow, m³/hr:			
oxygen	4.2-4.8	10-17	18.5-25
acetylene	3.5-7	8.5-14	15.5-21
Water flow, lit/hr . . .	200-400	500-700	800-1,000

Fig. 125 shows a flame-hardening torch using an oxy-kerosene flame. It likewise has several interchangeable tips *1* and incorporates an electrically heated kerosene evaporator *2* which is plugged into a low-voltage line by means of leads *3*. Cooling water is supplied by a tube *4*. The valve *5* adjusts the rate of oxygen flow

to the torch, the valve *6* adjusts the flow rate of liquid kerosene, and the valve *7* that of kerosene vapours to the mixing chamber.
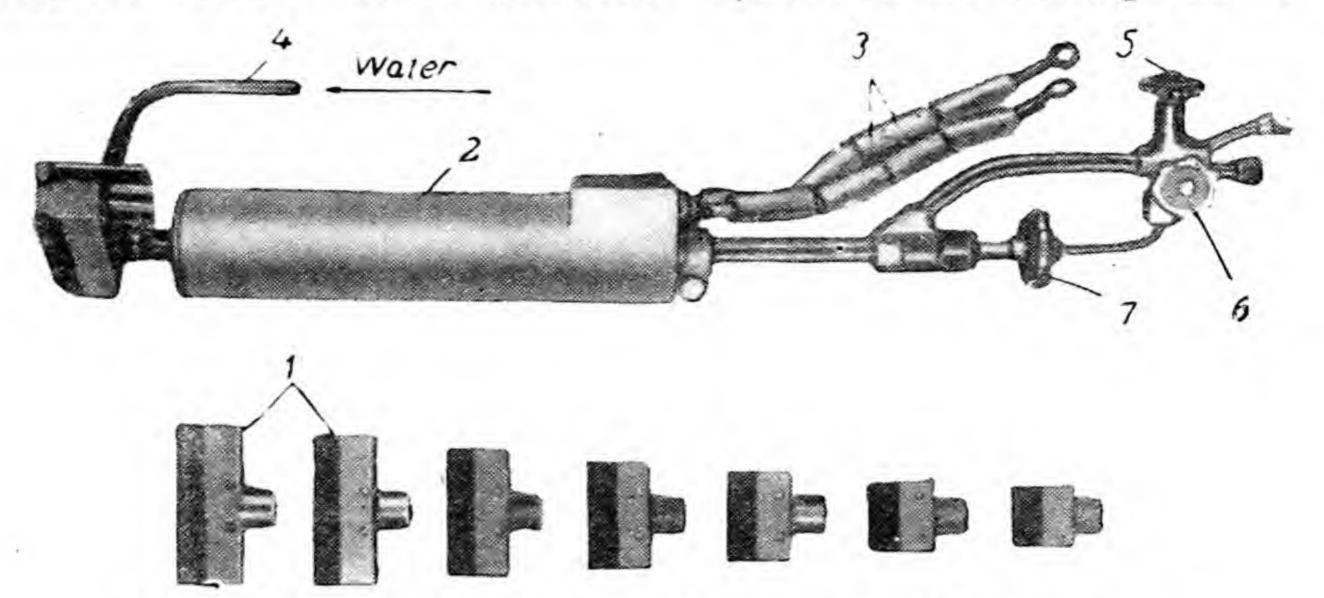

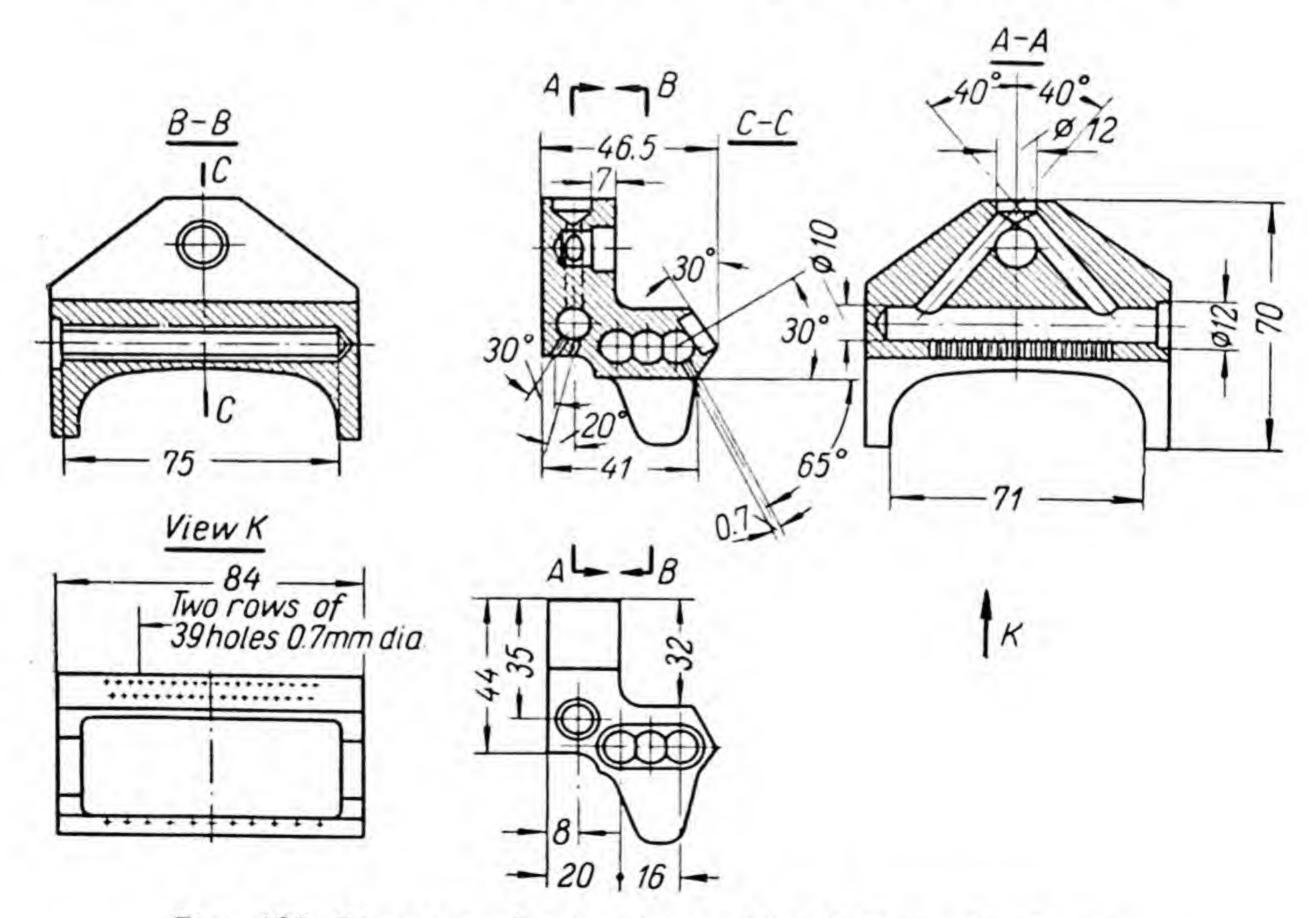Fig. 125. A flame-hardening torch using an oxy-kerosene flame

Fig. 126. Diagram of a tip for rail-head flame hardening

Parts of intricate shape are flame-hardened by means of tips more complex in outline. One of them, used to flame-harden rail heads, is shown in Fig. 126.

Oxygen cutting

THE PLACE OF OXYGEN CUTTING IN INDUSTRY

1. Definition and Classification

The process of oxygen cutting consists in burning the metal along a predetermined path and removing the resultant oxides from the cut with a jet of oxygen.

All oxygen cutting may be classed into three basic types. These are (a) flame cutting proper for severing; (b) flame machining in which grooves of circular cross-section are produced on the surface of the work; and (c) oxygen lance cutting.

A classification of flame cutting is presented in Fig. 127. In gas and flux cutting, the metal is continually preheated with a gas flame. In oxygen-lance cutting, a small spot on the piece of metal to be pierced is raised to a white heat, then the oxygen is turned on it.

2. Applications
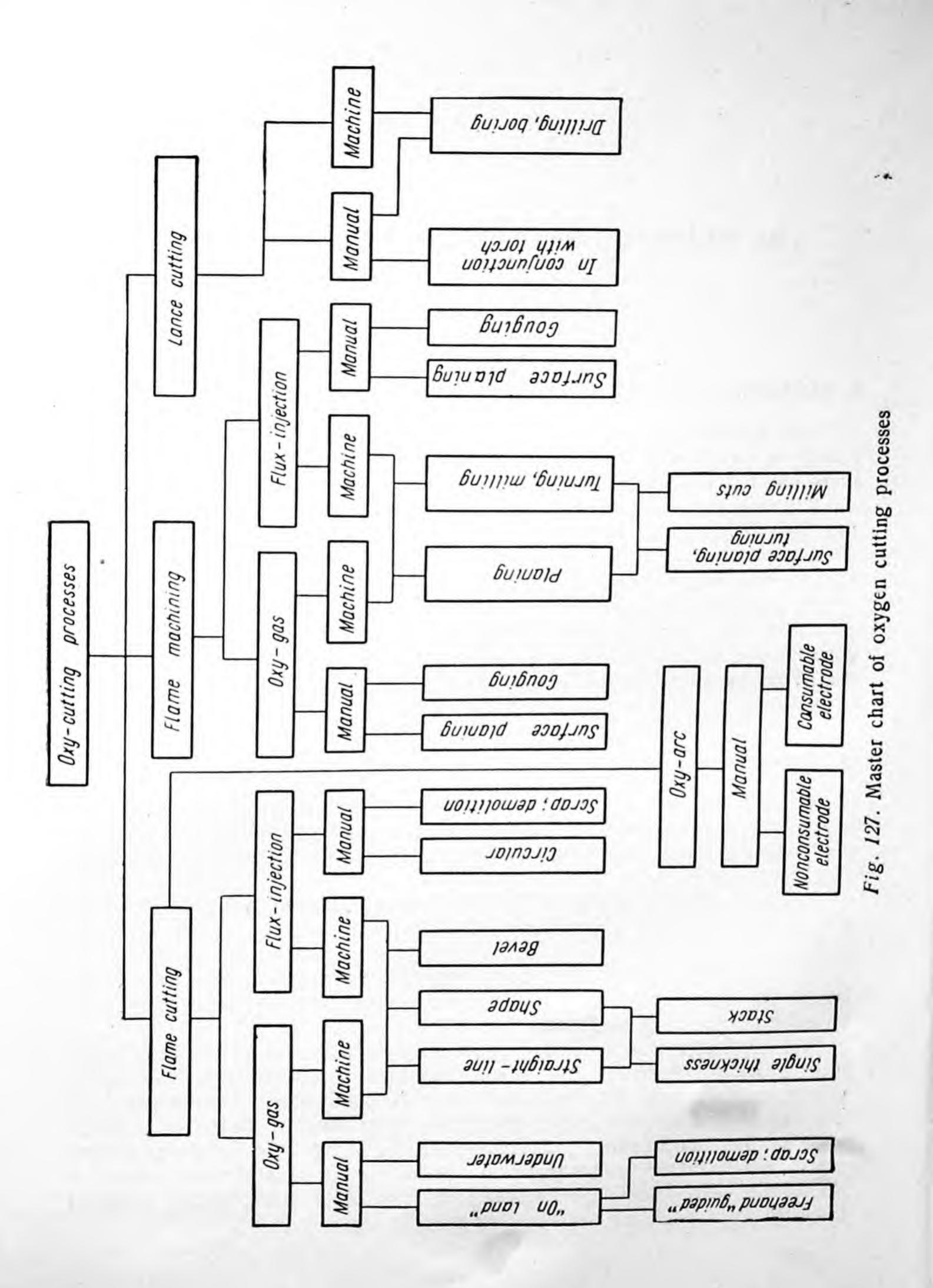
Flame cutting is used in almost any engineering industry, including metal-making, shipbuilding, boiler-making, locomotive and rolling-stock building, bridge construction, building construction, etc.

Widest use is being made of manual cutting which serves to cut steel plate and standard sections to shape, to make gussets, circles, flanges and other parts of simple or complex outlines.

Of late, the trend has been towards semiautomatic and automatic machine cutting. In the latter case, cutting machines are fitted with tracing devices.

As to flame machining, it can successfully complete with turning, milling, planing and other machining operations, especially where parts are to be produced on a limited scale. Flame machining has found widest use in the heavy engineering industry, especially in the manufacture of transport, hoisting and farming equipment where it is essential to make parts (chain-transmission sprockets, low-speed gears, etc.) to the final dimensions without leaving any stock for subsequent machining.

Fig. 127. Master chart of oxygen cutting processes

Flux cutting has found use in ferrous metallurgy, turbine building, refinery-equipment manufacture, electric metallurgy and some other industries and is applied to stainless and temperature-resistant steels, cast irons and copper alloys.

A few varieties of flame machining are employed for removing cracks or seams in billets, slabs and rounds en route to the various finishing departments of steel mills (deseaming); for removing superfluous metal in steel castings either purposely arranged or caused by defective conditions at the time of pouring (hogging); for bevelling the edges of the parts to be welded; for removing defective welds; and for some other purposes.

Oxygen-lance cutting has been employed to a limited degree. Its uses include drilling tap holes in furnaces, making deep holes in large masses of metal which are to be broken up by blasting, starting the bores of hollow shafts, initiating the kerf of odd-shaped parts in steel plate, etc.

THEORY OF OXYGEN CUTTING

1. The Nature of Oxygen Cutting

For its effect, flame cutting depends on the burning (intensive oxidation) of metal with a jet of oxygen, with the resultant oxides also removed by this oxygen jet out of the kerf.

The oxidation reaction for iron is as follows:

$$Fe + 0.5O_2 = FeO + 64.3 \text{ kcal/gram-mole}; \qquad (XX.1)$$

$$3Fe + 2O_2 = Fe_3O_4 + 266.9 \text{ kcal/gram-mole}; \qquad (XX.2)$$

$$2Fe + 1.5O_2 = Fe_2O_3 + 198.5 \text{ kcal/gram-mole}. \qquad (XX.3)$$

At elevated temperatures all of the three iron oxides may be produced in the cutting zone.

The technique of flame cutting resorts first to the preheating of a small area about the origin of the cut, until it attains a kindling temperature with oxygen. For steel, this is close to its melting point.

This preheating is accomplished by burning a fuel gas in oxygen. When the kindling temperature is attained, a jet of commercially pure oxygen (98-99. 7 per cent purity) is turned on. This jet of oxygen emerges from a central oxygen orifice in the cutting torch, called the cutting orifice, as the oxygen goes to burn and wash the metal out of the cut. There may be several small oxyacetylene flames arranged around the cutting orifice; they are called preheating flames, as they are used for that purpose.

Striking at the preheated area, the oxygen jet from the central orifice causes the surface metal to burn rapidly. The heat produced by the combustion raises the underlying layers to the kindling temperature so that the combustion extends through the entire thickness of the work. The molten oxides formed in the process are washed out of the cut by the eroding effect of the oxygen stream. As the molten metal runs down into the kerf, it comes in contact with the cooler metal, heats it and makes the cutting operation continuous.

Diagrammatically, flame cutting is shown in Fig. 128.

During flame cutting, especially in the case of its manual variety when the torch is directed against the metal ahead of the completed kerf, the oxides are continually removed from the kerf faces (Fig. 129), and the oxygen is in a more or less constant contact with the bare metal. Raised to kindling temperature, the metal can therefore be directly oxidised. Actually, the temperature in the kerf is above the melting point of steel. This is borne out by the presence of molten and unoxidised iron in the slag washed out of the kerf.
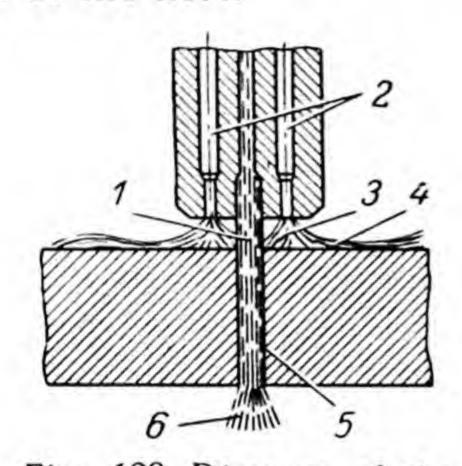
Fig. 128. Diagram of the cutting operation:
1 — cutting oxygen jet;
2 — oxy-fuel gas mixture;
3 — preheating flame cone;
4 — preheating flame streamer; *5* — kerf;
6 — oxides blown out by the oxygen jet
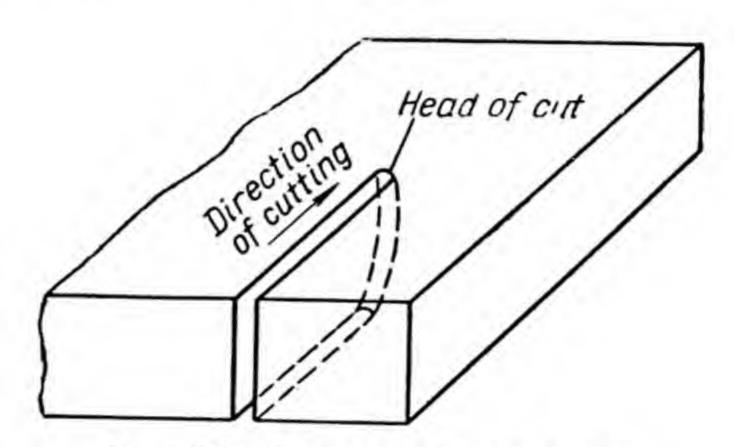
Fig. 129. Diagram of the kerf

Some of the metal is oxidised due to the diffusion of oxygen through the film of molten oxides present on the kerfface. It should be noted that in the case of a steady-state process which may only exist in ideal circumstances (with no vibration of the oxygen jet, uniform travel of the torch, etc.), the kerf face may be entirely covered by a thin film of molten oxides, and then diffusion may be the only way for oxygen to oxidise the metal.

2. Flame Cutability of Metals

For a metal to be successfully flame-cut, it should meet the following requirements.

First, its melting point should lie above its kindling temperature with oxygen.

By the kindling temperature of a metal is meant the point at which it is rapidly oxidised by the oxygen jet striking at its

surface. Investigators differ in their opinion as to the precise kindling temperature of ferrous metals. In all probability, this disagreement is due to variations in experimental techniques and in the purity of oxygen.

It seems safe to take this temperature equal to about 1350° C. Therefore, low-carbon steels with a melting point of about 1500° C meet the first requirement and are readily cutable. As the carbon content increases, the melting point of a steel goes down and its kindling temperature with oxygen rises, thereby reducing its cutability.

The slag formed in flame cutting differs in the percentage of unburnt metal. More unburnt metal will be present in ferrous metals higher in carbon. In cutting low-carbon steel, the slag contains about 15 per cent unburnt metal, while in the case of cast iron (containing over 1.7 per cent carbon), the slag is not in fact a slag at all, as it consists almost entirely of unoxidised metal and heavily burned impurities, washed out of the kerf.

Second, the oxides of the metal to be cut should melt at a point lower than the metal itself and below the temperature that is developed by cutting. Otherwise, the surface oxides will prevent any further combustion and removal of metal. This applies to high-chromium and chrome-nickel steels the surface oxides of which (Cr_2O_3) melt at about 2000° C; to aluminium alloys whose surface oxide (Al_2O_3) melts at about 2050° C, and to some other metals which cannot be flame-cut for this reason. Table 37 gives the melting points of the most common engineering metals and of their oxides.

Third, the heat produced by the combustion of the metal with a jet of oxygen must be sufficient to make the cutting process self-sustaining.

Table 37

Material	Melting point, °C	Material	Melting point, °C
Iron	1535	Brass	850-900
Low-carbon steel	about 1500	Tin bronze	850-950
High-carbon steel	1300-1400	Copper oxide	1236
Grey iron	1200	Aluminium	657
Ferrous oxide	1370	Aluminium oxide	2020-2050
Ferrous-ferric oxide	1527	Zinc	419
Ferric oxide	1565	Zinc oxide	1800
Copper	1083		

Note: For alloys, these are but approximate figures

In cutting low-carbon steel sheet, the combustion of iron and impurities supplies about 70 per cent of the total heat, the balance being produced by the preheating flame.

The quantity of heat produced by the oxidation reactions of some of the metals is tabulated in Table 38.

Table 38

Oxide	Heat of oxidation, kcal/gram-mole	Oxide	Heat of oxidation, kcal/gram-mole
FeO	64.3	MnO	93.1
Fe_3O_4	266.9	Mn_2O_3	232.7
Fe_2O_3	198.5	Mn_3O_4	336.5
CuO	37.5	WO_2	130.5
Cu_2O	40.6	WO_3	195.7
Al_2O_3	393.3	V_2O_3	320
ZnO	83.36	MoO_2	130
SnO	67.9	MoO_3	184
NiO	58.4	PbO	52.36-51.96
Cr_2O_3	273		(red) (yellow)

Fourth, the metal to be flame-cut must not have excessive thermal conductivity, or the runaway of heat may prevent the cutting operation from starting at all or may cause it to interrupt frequently. This applies to copper, aluminium and their alloys.

Fifth, the oxides formed in cutting should be fluid when molten. Otherwise, the molten slag will be difficult to remove from the kerf, and the progress of the cutting operation will be hindered. This applies to cast iron the cutting of which produces slag high in silicon oxide (SiO_2) which is highly refractory and has a broad temperature range of viscosity.

Sixth, for a metal to be readily flame-cut, it should have a limited content of the impurities which adversely affect its cutability (C, Cr, Si, etc.) and raise its hardenability (Mo, W, etc.).

Summing up, it is obvious that the best cutable materials are pure iron and low-carbon steel. With carbon running more than 0.7 per cent, flame cutting calls for a preheat of 300 to 650° C and becomes impossible with a carbon content of over 1-1.2 per cent.

Normally, cast iron cannot be flame-cut. The reasons for this are that its kindling temperature with oxygen is above its melting point (which, for grey iron, is about 1200° C) and that the refractory silicon oxide (SiO_2) makes the slag insufficiently fluid to be

removed by the oxygen jet from the kerf. Last but not least, too much carbon monoxide and dioxide is formed, contaminating the cutting oxygen jet and reducing the efficiency of combustion.

High-chromium and chrome-nickel steels cannot likewise be flame-cut by any normal technique due to the refractory chromium oxide formed on the surface of the metal.

Nonferrous metals, especially copper, aluminium and their alloys have low flame cutability because of refractory oxides, marked heat conductivity, and low heat of oxidation (which especially applies to copper and its alloys).

3. Preheating

A flame cut is started by heating a small area around the start of the cut until it attains a kindling temperature with oxygen. This preheating may be accomplished by any means capable of raising the steel work to 1300 or 1350° C within the shortest possible stretch of time. The most commonly used method of preheating is with an oxy-acetylene flame, although it is equally possible to use an electric arc, induction heating, etc. The factors affecting the efficiency of heating are as follows.

(1) The Fuel Gas for and Heat Input from the Preheating Flame. Any of the gases presented in Table 9, Chapter IV, may be used for preheating purposes. As in welding, acetylene has found widest use in cutting compared with other gases.

The advantages of an oxy-acetylene preheating flame are higher effective heat input (see Fig. 65) and flame temperature (see Table 9, Chapter IV) in comparison with other gas flames. All this speeds up the preheating with an oxy-acetylene flame. On the other hand, gases other than acetylene are cheaper to produce and easier to obtain.

Before the metal is kindled to burn with oxygen, the only source of the heat supplied is the preheating flame. Therefore, the duration of actual preheating depends on the fuel used.

For an oxy-acetylene flame, this stretch of time is approximately as follows:

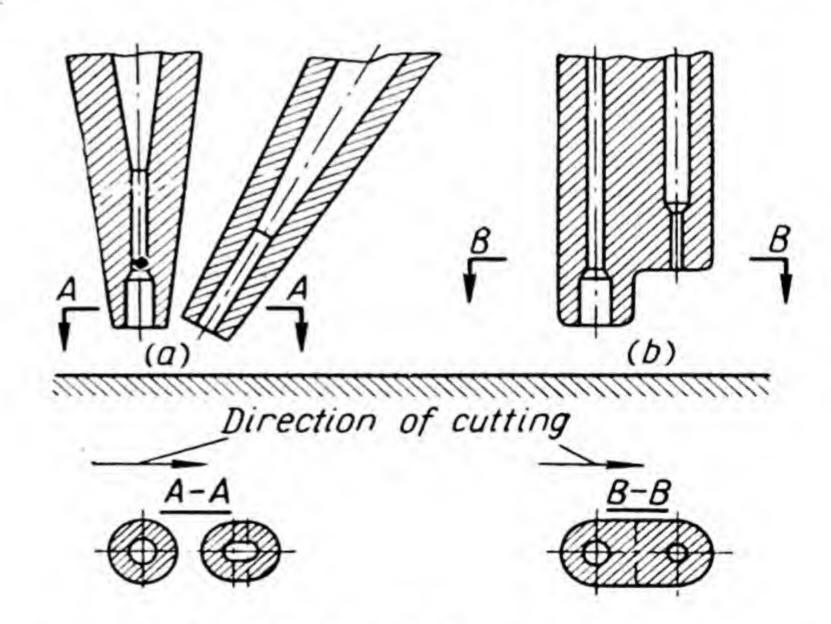| Steel thickness, mm	Duration of preheat, sec
10-20	5-10
20-100	7-25
100-200	25-40

In the case of other fuel gases of lower calorific value and a lower maximum flame temperature, this time is longer for the same effective heat input.

After kindling temperature is attained, the cutting operation proceeds freely and regularly, the heat to sustain it being mainly supplied by the combustion of the metal. According to some investigators, the preheating flame accounts but for 28 per cent of the total heat generated in cutting low-carbon steel plate 15 mm thick.

While theoretically preheating should not be necessary beyond that of starting the cut, in practice it is maintained throughout the cutting operation — an expedient which ensures continuous cutting. Otherwise, cutting will be immediately stopped, as the oxygen jet cannot kindle the cooler metal ahead of the cut for the reason that some heat is lost through conductivity, radiation and convection.

(2) The Shape and Arrangement of Preheating Nozzles Around the Cutting Oxygen Orifice. As in the case of the welding flame, the oxy-acetylene preheating flame in cutting is produced

Fig. 130. Arrangement of the cutting oxygen orifice and the preheating tip

by a fuel gas burning in oxygen. As distinct from the welding flame, however, the preheating flame is produced either by a tip having a continuous annular slot (as in manual cutting torches, Fig. 59a) or by a multiflame tip with several cylindrical nozzles arranged all the way round a circle (as in most cutting machines, Fig. 59b). Multiflame tips give a more stable preheating flame than the annular slot type. This is especially true of gases of low calorific value (such as city, natural or shale gases).

The most efficient preheating is obtained with multiflame tips in which the nozzles converge towards the axis of the cutting oxygen orifice, thereby reducing the area to which the preheating flames are applied and concentrating the heat input.

Sometimes, cutting torches have a preheating flame arranged in line with and ahead of the cutting oxygen orifice (Fig. 130*a* and *b*). With this arrangement, the preheat is applied in a narrow path, which is especially essential in cutting light gauges, but only straight cuts can be made.

In both the annular slot type and multiflame tips, the preheating flame or flames are arranged all the way round the cutting oxygen orifice, convenient for shape cutting. A drawback of the annular slot type of torch (see Fig. 59*a*) is that fuel consumption is greater than with the multiflame type. In general, the latter type (Fig. 59*b*) is more efficient on many counts.

(3) The Tip-to-Metal Distance. The time necessary to raise the workpiece to kindling temperature greatly depends on the distance from the tip to the metal surface.

The flame is hottest within 1.5 to 2 mm of its inner cone. Therefore, in preheating the torch should be positioned so as to maintain this distance, without making the inner cone touch the surface. Otherwise, the cut faces may be carburised. The precise separation between the tip and the work is of special importance in the case of light gauges.

(4) The Oxygen-to-Gas Ratio. The preheating flame may be either normal or oxidising; the flame with an excess of fuel gas should not be used, as it may carburise the metal.

The recommended oxygen-to-gas ratios are given in Sec. 4, Chapter VIII.

The composition of the preheating flame should be adjusted with the cutting oxygen turned on. The point is that oxygen for both the cutting orifice and the preheating flame comes over one and the same hose; if the preheating flame were adjusted with the cutting oxygen flow shut off, it would be oxygen-impoverished in the course of cutting, and the metal would be carburised due to the excess of fuel gas.

As has been reported by some investigators, the excess of oxygen desirable in the preheating flame depends on the specific rate of fuel-gas flow (or the thickness of the metal to be cut), and should be smaller for higher flow rates. For a rate of acetylene flow of 0.15 m^3/hr, the maximum effective heat input at the lowest consumption of both gases will be obtained with an oxygen-to-gas ratio of 1.7. For a rate of acetylene flow of over 0.65 m^3/hr, the best results are obtained with an oxygen-to-gas ratio of 1.1.

(5) The Cleanliness of the Metal Surface. The presence on the metal surface of mill scale, sand, slag or any other contaminants impedes direct contact between the metal and the flame. In many cases, dirt on the metal surface prevents the cut from being started or continued uninterruptedly, or the speed of cutting is reduced although the preheating flame may be very hot.

(6) The Cooling Effect of the Cutting Oxygen Jet. The oxygen used for cutting may grow too cold due to a variety of factors. These are: (a) the oxygen is withdrawn at an excessively high rate; (b) the oxygen is throttled or wiredrawn in the pressure regulator too much; or (c) the oxygen is severely throttled as it issues from the tip. Approximately, as the oxygen issues from the tip and into the atmosphere, throttling causes it to cool by as much as $1/4°$ C per atmosphere of pressure drop. To put it differently, the oxygen is throttled and cooled more at the higher working pressures and has a greater cooling effect on the metal being cut.

It should be noted, however, that in cutting light and medium gauges of steel, this cooling effect may only retard the speed of cutting, but is unable to prevent the cut from being started or continued uninterruptedly, for the preheating flame compensates for the cooling effect.

The cooling effect may be moderated and the rate of cutting improved by using several oxygen cylinders connected to a common manifold where the rate of oxygen flow is high; or by employing a two-stage regulator; or by passing the cutting oxygen jet through a coiled tube immersed in a hot-water bath or heated electrically.

The former two expedients are not difficult to employ and are therefore widely used in cutting practice. As to heating the cutting oxygen jet, this technique has been employed on a limited scale, as it is prohibitively complicated to apply and is of low efficiency.

4. The Material and Heat Balance of Steel Cutting by the Oxy-Acetylene Flame

Analyses of cutting slags have yielded widely differing results. Some investigators have detected the three iron oxides, others only FeO and Fe_3O_4, others have found FeO and Fe_2O_3; still others have traced only Fe_3O_4. Yet, they all agree that cutting slag contains some percentage of unburnt metal removed from the kerf by the eroding action of the oxygen jet.

Investigators likewise differ in their data on the oxide percentage in cutting slag (Table 39). A very likely cause of the

discrepancy may be the various test techniques employed, the differing cutting conditions used and the inaccuracy of chemical analysis.

Table 39

Content of Iron and Its Oxides in Cutting Slag

Composition, per cent				Total, per cent	Year of publication
Fe	FeO	Fe_3O_4	Fe_2O_3		
4.6-23	47.1-57.8	—	21-46.1	77-95.4	1933
40	—	60	—	60	1936
15	50	35		85	1936
30-40	—	60-70	—	60-70	1937
20	—	80	—	80	1941
14	66	—	20	86	1949
9.4	27.2	63.4	—	90.6	1952
4.2-39.9	18-51.3	19.4-77.8	—	70.7-95.8	1959

Nor can the chemical analyses of cutting slags be identical, because the iron-oxygen system is remarkable for the fact that FeO and Fe_3O_4, while forming solid and liquid solutions in each other, also produce solid and liquid solutions in iron, thereby absorbing, as it were, oxygen continually.

A hypothesis has been advanced that in conditions of gas cutting, when the slag has a temperature of over 1383° C, the Fe_2O_3 formed in it should change to Fe_3O_4 in the air, releasing some oxygen. According to Rostovtsev, this is owing to the fact that at 1383° C Fe_2O_3 has a dissociation pressure of 0.21 atm (gauge), which is as high as the partial pressure of atmospheric oxygen. It should be noted that this hypothesis may hold principally in the case of light-gauge steel, when little slag is formed, and all of it is in contact with ambient air. In the cutting of heavy-gauge steel much slag is formed; flowing out of the kerf, it produces considerable accumulations in which the inside material is shut off from the atmosphere.

Another important possibility is that after it is washed out of the kerf, slag cools and can be oxidised to Fe_2O_3 by atmospheric oxygen.

From the viewpoint of the heat balance of cutting the final state of the material removed from the kerf is of minor importance. What is important to know is the composition of the oxides in the cutting zone, i. e., before the slag is removed from the cut. Any direct determination of oxide composition at this

moment is out of the question. Still, it is a known fact that during the cutting operation the three iron oxides should be formed according to reactions (XX. 1) through (XX. 3).

By averaging the data from Table 39, the composition of slag may roughly be determined as having 25-35 per cent FeO, 35-45 per cent Fe_3O_4, 5-15 per cent Fe_2O_3, and 15-25 per cent Fe.

It should be taken into account, however, that very little, if any, of the highest oxide is found in cutting slag, and also that the heats of the reactions producing Fe_3O_4 and Fe_2O_3 are approximately comparable. For this reason, it is safe to base both the material and heat balance of flame cutting on FeO and Fe_3O_4. The latter should be taken equal to the sum of Fe_3O_4 and Fe_2O_3 in cutting slag. Therefore, cutting slag may be assumed to have 20 per cent Fe, 30 per cent FeO, and 50 per cent Fe_3O_4.

(1) The Material Balance. Iron burns to form FeO as follows:

$$2Fe + O_2 = 2FeO; \quad \left. \begin{array}{l} \\ 2 \times 55.85 + 32 = 143.7 \end{array} \right\} \qquad \text{(XX.4)}$$

Assuming that 30 per cent of the iron burns to form FeO, we obtain

$$\frac{0.3 \times 111.7}{111.7} + \frac{0.3 \times 32}{111.7} = \frac{0.3 \times 143.7}{111.7},$$

or, in parts by weight:

$$0.3Fe + 0.086O_2 = 0.386FeO. \qquad \text{(XX.5)}$$

To form Fe_3O_4, iron burns to the following reaction:

$$3Fe + 2O_2 = Fe_3O_4; \quad \left. \begin{array}{l} \\ 3 \times 55.85 + 2 \times 32 = 231.55 \end{array} \right\} \qquad \text{(XX.6)}$$

Assuming that 50 per cent of the iron burns to form Fe_3O_4, we obtain:

$$\frac{0.5 \times 167.55}{167.55} + \frac{0.5 \times 64}{167.55} = \frac{0.5 \times 231.55}{167.55},$$

or, in parts by weight:

$$0.5Fe + 0.191O_2 = 0.691Fe_3O_4. \qquad \text{(XX.7)}$$

The total amount of iron burned to FeO and Fe_3O_4 plus that part of iron which remains unburnt is obtained by adding together Eq. (XX. 5) and (XX. 7) plus 0.2 part of Fe (by weight), or:

1 part Fe $+ 0.277$ part $O_2 = 1.277$ parts FeO, Fe_3O_4 and Fe. (XX.8)

In other words, theoretically it takes 0.277 part (by weight) of oxygen 100-per cent pure to remove one part (by weight) of iron from the kerf.

The quantity of oxygen necessary to burn 1 cu cm of Fe can be found by Eq. (XX. 3):

$$V_{ox} = \frac{7.86 \times 0.277}{1.33} = 1.64 \text{ lit/cm}^3 \text{ Fe,} \qquad (XX.9)$$

where V_{ox} is the specific consumption of oxygen in litres per cu cm of iron; 7.86 is the specific gravity of pure iron and 1.33 is the specific gravity of oxygen at 20° C and 760 mm Hg.

The actual consumption of oxygen is a somewhat different value which varies with the purity of cutting oxygen and some other factors which are difficult to asses quantitatively, such as the suction of ambient air by the oxygen jet, the shape and velocity of the oxygen jet, metal thickness, steel analysis, etc.

In actual cutting, some more oxygen is spent in the preheating flame and to wash the oxides out of the kerf.

Furthermore, less oxygen is consumed per unit thickness on heavier gauges, because the oxygen jet is better utilised in the kerf.

When the theoretical consumption of cutting oxygen and the resultant slag composition are known, it is possible to draw a balance of the cutting operation, either per 1 cu cm or 1 gram of the iron burnt, or per unit length of the kerf (1 metre or 1 centimetre).

For oxy-acetylene cutting this can be written as folows:

$$G_{Fe} + G_{ph\ ox} + G_{c\ ox} + G_{ox\ b} + G_a = G_{FeO} + \qquad (XX.10)$$

$$+ G_{Fe_3O_4,\ Fe_2O_3} + G'_{Fe} + G_{CO_2} + G_{H_2O} + G_{O_2} \dots,$$

where G_{Fe} = weight of iron removed from the kerf;
$G_{ph\ ox}$ = weight of preheating oxygen;
$G_{c\ ox}$ = weight of cutting oxygen;
$G_{ox\ b}$ = weight of oxygen spent to blow out molten metal out of the kerf;
G_a = weight of preheating acetylene;
G_{FeO} = weight of resultant FeO;
$G_{Fe_3O_4,\ Fe_2O_3}$ = weight of resultant Fe_3O_4 and Fe_2O_3;
G'_{Fe} = weight of iron removed by the cutting oxygen jet out of the kerf unburnt;
G_{CO_2} = weight of resultant CO_2;
G_{H_2O} = weight of resultant water vapour;
G_{O_2} = weight of oxygen unused in combustion.

The values of the terms in the above equation can be found as follows.

The specific consumption of cutting oxygen ($G_{c\,ox}$) is found by calculation; the value thus found is then subtracted from the aggregate specific consumption of cutting oxygen as shown by the rotameter; this gives the specific oxygen consumption to blow molten metal out of the kerf ($G_{ox\,b}$) and oxygen losses (G_{O_2}).

The specific consumption of preheating oxygen ($G_{ph\,ox}$) and acetylene (G_a) is also read from the respective rotameters.

Proceeding from the slag analysis, the resultant amounts of FeO and Fe_3O_4 (G_{FeO} and $G_{Fe_3O_4}$) are obtained by calculation on the basis of the specific consumption of cutting oxygen. It should be remembered that it takes 0.223 gram of O_2 to produce one gram of FeO and 0.276 gram of O_2 to produce one gram of Fe_3O_4.

The amount of iron removed from the kerf unburnt (G'_{Fe}) depends on the slag composition assumed. The remaining two terms (G_{CO_2} and G_{H_2O}) are found as the difference between the right- and left-hand sides of Eq. (XX. 10).

(2) The Heat Balance. The *sources of heat* in flame cutting are the preheating flame and the combustion of iron and impurities with oxygen.

The heat supplied by the preheating flame is equal to the low heat value of the fuel gas, assuming that it is fully burned with oxygen. The low heat value of acetylene at 20° C and 760 mm Hg is 12,600 cal/lit. The amount of heat per linear metre of the kerf will be:

$$H_{ph} = \frac{3.5V_a}{\omega} \text{ cal per linear metre,} \qquad (XX.11)$$

where $3.5\,V_a = \dfrac{12,600 V_a}{3,600}$, which is the amount of heat per second released by the flame when acetylene is fully burned with oxygen, as expressed through the hourly rate of acetylene flow V_a in lit/hr and the low heat value of acetylene in calories; $\overline{\omega}$ is the cutting speed in metres/min.

The heat produced by iron burning to FeO and Fe_3O_4 is the sum of the heats given up by the reactions described by Eq. (XX. 1) and Eq. (XX. 3). For one gram of iron, this will be 1,150 cal for FeO and 1,580 cal for Fe_3O_4.

Then, for a given amount of iron burning to FeO and Fe_3O_4, the heat per linear metre of the cut will be:

$$H_{i\,c} = 7.8 \times 100bs \,(r_{FeO} \times 1,150 + r_{Fe_3O_4} \times 1,580) \text{ cal} \quad (XX.12)$$

per linear metre,

where $H_{i\,c}$ is the heat of iron combustion per linear metre; 7.8 is the density of low-carbon steel in gram/cm³; $100bs$ is the

volume of burnt iron in cubic centimetres per linear metre; b is the width of the cut in centimetres; s is the metal thickness in centimetres; and r_{FeO} and $r_{Fe_3O_4}$ are the iron parts (by weight) burning to FeO and Fe_3O_4, respectively.

The heat supplied by the combustion of the impurities in steel, mainly C and Mn, is as insignificant as their percentage, and in plain low-carbon steel does not approximately exceed 3.5 per cent of the heat produced by the combustion of the iron, or $H_{i\,c}$, i. e.,

$$H_{imp\,c} = 0.035 H_{i\,c}. \qquad (XX.13)$$

The heat available in gas cutting is spent (A) to heat the metal; (B) to heat the slag; (C) as losses (a) with the unused proportion of oxygen in the cutting jet, (b) with waste gases in the preheating flame, and (c) through radiation.

The heat spent to heat one linear metre of metal, $H_{m\,h}$, is expressed in calories per linear metre and is the difference of the heat content of the steel plate before and after cutting, S_0 and S, respectively:

$$H_{m\,h} = S - S_0. \qquad (XX.14)$$

The heat content of the metal prior to cutting depends on the ambient temperature, T_0, and is found by the equation

$$S_0 = G \times c_0 \times T_0,$$

where G is the weight of one linear metre of steel plate, and c_0 is the heat capacity of steel at T_0.

After cutting, the heat content of the steel plate is usually determined by calorimetry. It should be noted, however, that the heat content of the plate cut into pieces is always somewhat greater than that of a whole plate, as there is always some slag left on the cut faces, not removed by the oxygen jet. The amount of heat imparted to the metal by the slag may be taken equal to one-tenth of what the slag receives itself, or $H_{mh}^{sl} = 0.1\ H_{sl\,h}$.

The heat absorbed by the metal during the cutting operation comes as part of the heat produced by the preheating flame and as part of the heat produced by the combustion of the iron.

For its effect the preheating flame mainly depends on the convective and radiant heat exchange between the flame envelope and the metal adjacent to the cut. The heat imparted to the metal propagates in the plate due to thermal conductivity.

The heat produced by the combustion of the iron is mainly imparted to the plate through thermal conductivity, as the kerf faces come in contact with the products of iron combustion and,

partly, with the slag flowing down the kerf. The radiant heat exchange between the products of iron combustion and the kerf faces plays a very modest part. The heat imparted to the metal likewise propagates throughout the plate due to thermal conductivity.

As the cutting oxygen jet goes through the preheating flame, the flame picks up some more oxygen, and the effective heat input from it increases (see Fig. 70*b*). In addition, the excess of oxygen reduces the size of the inner cone, intermediate zone and envelope, thereby concentrating the heat of the flame near its inner cone and raising the efficiency of metal heating.

At the same time, some of the heat is imparted to the cutting oxygen jet, which fact reduces the heat input from the flame. Unfortunately, we lack any quantitative data on the heat loss due to the heating of the cutting oxygen jet. Nor can we say for certain which of the two factors affects the heat input from the flame more. Assuming that neither of them affects the effective heat input, the heat imparted by the preheating flame to every metre of the cut, H_{mh}^{ph}, can be found by the equation

$$H_{m\,h}^{ph} = \frac{60q}{\omega},\qquad\text{(XX.15)}$$

where q is the effective heat input from the flame in cal/sec (see Sec. 4, Chapter VIII) and ω is the cutting speed in metres/min.

Then the proportion of heat produced by the iron burning to FeO and Fe_3O_4 and spent to heat the metal will be

$$H_{m\,h}^{i\,c} = H_{m\,h} - H_{m\,h}^{ph},\qquad\text{(XX.16)}$$

where $H_{m\,h}$ is the total heat imparted to the metal, and H_{mh}^{ph} is the amount of heat imparted to the metal by the preheating flame and equal to $\frac{60q}{\omega}$.

The total amount of heat going to raise the plate to kindling temperature is

$$H_{m\,h} = H_{m\,h}^{ph} + H_{m\,h}^{i\,c} + H_{m\,h}^{sl}.\qquad\text{(XX.17)}$$

The heat imparted to the slag for evely linear metre of the kerf, $H_{sl\,h}$, in calories per linear metre, is the difference between the heat contents of the cutting slag, S'_{sl}, at T' and of the iron, S_{Fe}, at the temperature of the plate prior to cutting, or

$$H_{sl\,h} = G\,(S'_{sl} - S_{Fe}),\qquad\text{(XX.18)}$$

where G is the weight of the iron in grams in the kerf one metre long.

As cutting slag mainly consists of FeO, Fe_3O_4 and small quantities of unburnt iron, the heat content of slag, S'_{sl}, can be regarded as the sum of the heat contents of its components. Then Eq. (XX. 18) may be thrown into another form:

$$H_{sl\,h} = G\left[(r_{FeO} \times S'_{FeO} + r_{Fe_3O_4} \times S'_{Fe_3O_4} + r_{Fe} \times S'_{Fe}) - S_{Fe}\right], \quad (XX.19)$$

where S'_{FeO}, $S'_{Fe_3O_4}$ and S'_{Fe} are heat contents of FeO, Fe_3O_4 and Fe at the temperature of the slag, T';

r_{FeO}, $r_{Fe_3O_4}$, r_{Fe} are parts (by weight) of FeO, Fe_3O_4 and Fe, respectively

In most cases, however, the steel prior to cutting is at about room temperature, and the heat content of the iron, S_{Fe}, may be neglected. Then Eq. (XX. 19) may be rewritten as follows:

$$H_{sl\,h} = G\,(r_{FeO} \times S'_{FeO} + r_{Fe_3O_4} \times S'_{Fe_3O_4} + r_{Fe} \times S'_{Fe}). \quad (XX.20)$$

The temperature of slag in cutting is about $1600°\,C$ *. At this temperature the heat content of iron is 340 cal/gram, but we lack reliable data about the heat content of FeO and Fe_3O_4 at this temperature. If we neglect that the oxides are in the molten state, their heat content may be derived from their average specific heat capacities, which, according to N. N. Klebanov, are 0.16 cal/°C for FeO and 0.24 cal/°C for Fe_3O_4. Then the heat content of FeO will be:

$S'_{FeO} =$ (specific heat capacity of FeO at T') + heat of fusion of
  FeO ** = $(0.16 \times 1,600) + 50 = 306$ cal/g (or approximately 310 cal/g)

and of Fe_3O_4:

$S'_{Fe_3O_4} = (0.24 \times 1,600) + 50 = 434$ cal/g (or approximately 430 cal/g).

Substituting these values in Eq. (XX. 20) and neglecting the heat content of the iron, S_{Fe}, at the temperature of the plate prior to cutting, the heat absorbed by the slag as it is raised to $1600°\,C$ will be

$$H_{sl\,h} = G\,(r_{FeO} \times 310 + r_{Fe_3O_4} \times 430 + r_{Fe} \times 340).$$

Referred to one linear metre of the cut, the weight of the burnt metal will be $G = 7.8 \times 100\,bs$, where 7.8 is the density of plain low-carbon steel in g/cm³. Then Eq. (XX. 20) may finally

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* This figure has been obtained at the Autogenous Welding Research Institute by means of an optical pyrometer.
  ** The heat of fusion for both oxides (FeO and Fe_3O_4) is 50 cal/g.

be rewritten as follows:

$$H_{sl\ h} = 7.8 \times 100bs\,(r_{FeO} \times 310 + r_{Fe_3O_4} \times 430 + r_{Fe} \times 340). \quad (XX.21)$$

The heat losses, H_{loss}, (a) to heat the unused oxygen in the cutting jet; (b) with the waste gases of the preheating flame; and (c) due to radiation into surroundings can be found as the difference between the available heat and the heat spent to raise the metal and slag in temperature, or

$$H_{loss} = (H_{ph} + H_{i\ c} + H_{imp\ c}) - (H_{m\ h}^{ph} + H_{m\ h}^{i\ \&\ imp\ c} + H_{sl\ h}). \quad (XX.22)$$

In a general form, the heat balance of flame cutting will be as follows:

$$H_{ph} + H_{ic} + H_{imp\ c} = H_{m\ h}^{ph} + H_{m\ h}^{i\ \&\ imp\ c} + H_{sl\ h} + H_{loss}. \quad (XX.23)$$

Example: Find the heat balance of the cutting operation for plate 15 mm thick in plain low-carbon steel.

The cutting speed $\omega = 0.53$ m/min; the acetylene flow rate in the preheating flame $V_a = 460$ lit/hr; the width of the cut $b = 0.3$ cm; the slag contains 0.3 FeO, 0.5 Fe_3O_4 and 0.2 Fe (parts by weight).

I. Available Heat

1. The heat supplied by the preheating flame, Eq. (XX. 11):

$$H_{ph} = \frac{460 \times 3.5 \times 60}{0.53} = 182,000 \text{ cal/linear metre.}$$

2. The heat of combustion of the iron, Eq. (XX. 12):

$$H_{i\ c} = 7.8 \times 100 \times 0.3 \times 1.5\,(0.3 \times 1,150 + 0.5 \times 1,580) = 398,000 \text{ cal/linear metre.}$$

3. The heat of combustion of the impurities, Eq. (XX. 13):

$$H_{imp\ c} = 0.035 \times 398,000 = 14,000 \text{ cal/linear metre.}$$

II. Heat Spent

1. The heat of the preheating flame spent to raise the metal to kindling temperature, Eq. (XX. 15):

$$H_{m\ h}^{ph} = \frac{60 \times 880^*}{0.53} = 100,000 \text{ cal/linear metre.}$$

2. The heat imparted to the slag, Eq. (XX. 21):

$$H_{sl\ h} = 7.8 \times 100 \times 0.3 \times 1.5\,(0.3 \times 310 + 0.5 \times 430 + 0.2 \times 340) = \\ = 130,000 \text{ cal/linear metre.}$$

3. The heat transferred to the metal by the slag left on the kerf faces (10 per cent of the heat content of the slag):

$$H_{m\ h}^{sl} = 0.1 H_{sl\ h} = 0.1 \times 130,000 = 13,000 \text{ cal/linear metre.}$$

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* The effective heat input, q, from the preheating flame of an annular multiflame tip (Fig. 59b) is taken equal to 880 cal/sec. Then the rate of acetylene flow will be $V_a = 460$ lit/hr from the chart in Fig. 65 for $\beta_0 = 1.15$.

4. The total heat absorbed by the metal.

According to Yevseyev and Shorshorov, calorimetric measurements have shown that $H_{m\,h}^{ph}$ accounts for about 30 per cent of the total heat absorbed during the cutting operation on a low-carbon steel plate 15 mm thick. Then, the total heat will be:

$$H_{m\,h} = \frac{H_{m\,h}^{r\,h}}{0.3} = \frac{100,000}{0.3} = 333,000 \text{ cal/linear metre.}$$

5. The part of the heat of combustion of the iron and impurities absorbed by the metal:

$$H_{m\,h}^{i\,\&\,imp\,c} = H_{m\,h} - H_{m\,h}^{ph} - H_{m\,h}^{sl} = 333,000 - 100,000 - 13,000 =$$
$$= 220,000 \text{ cal/linear metre.}$$

6. The losses of heat to waste gases, radiation and to the unused oxygen in the cutting jet:

$$H_{loss} = (182,000 + 398,000 + 14,000) - (100,000 + 220,000 + 130,000) =$$
$$= 594,000 - 450,000 = 144,000 \text{ cal/linear metre.}$$

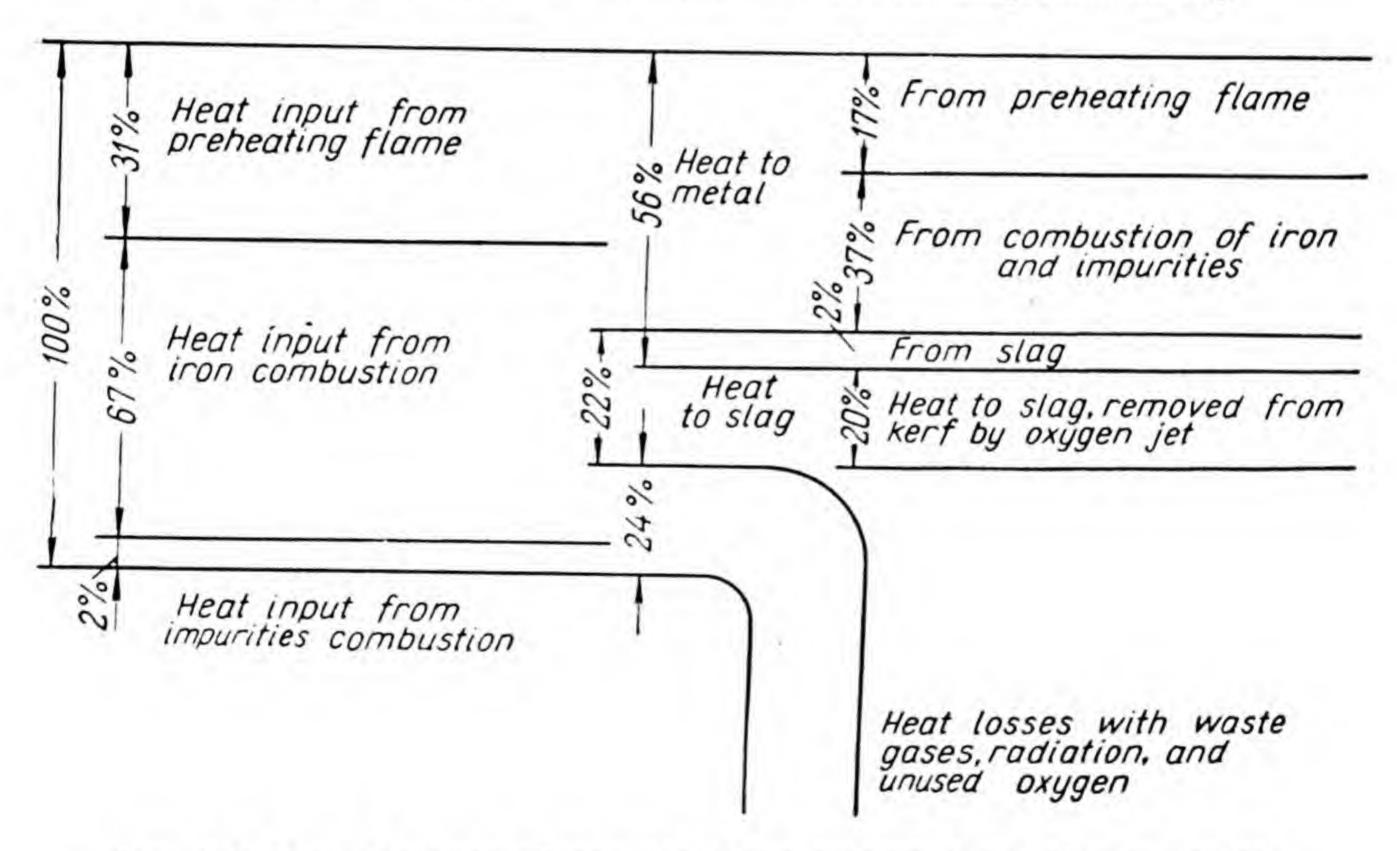Fig. 131 shows the above heat distribution diagrammatically.

Fig. 131. Heat balance of oxy-acetylene cutting (steel 15 mm thick)

The items of the heat balance in gas cutting may, of course, vary in magnitude for various metal thicknesses and cutting conditions.
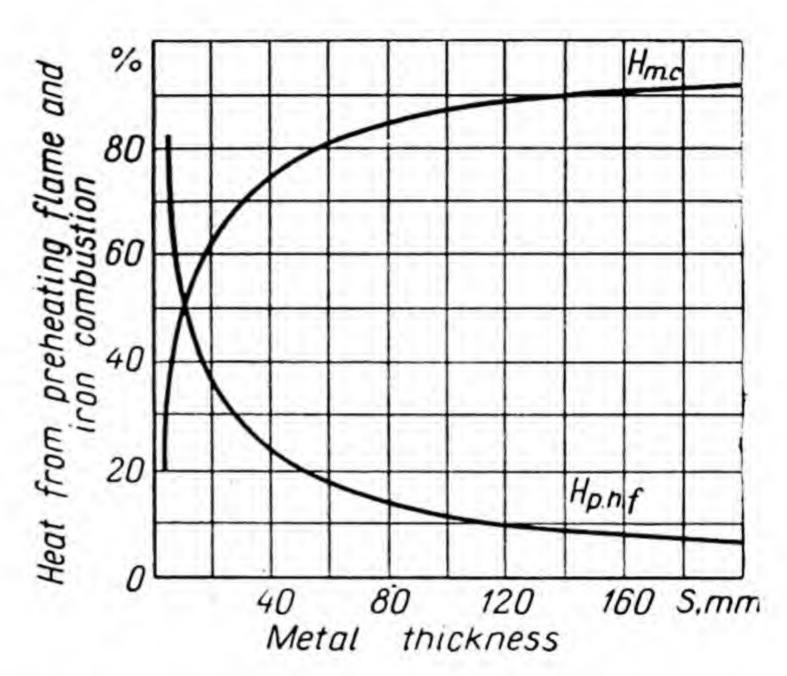
This is borne out by the equations given and the diagram presented in Fig. 132, which is based on recent studies. It should be

added that the experimental values of the various items have been very close to those derived from the equations.

It is characteristic of flame cutting that as the thickness of steel is progressively decreased (down to 10 or 12 mm), the preheating flame becomes the principal source of heat and not the heat of oxidation as in the case of heavy gauges (see Fig. 132).

Fig. 132. The quantity of heat imparted to the workpiece by the preheating flame and the combustion of iron related to metal thickness

5. The Effect of Impurities on Cutting and the Hardenability of the Kerf Faces

The impurities contained in steel have an appreciable effect on the cutting operation. Some of them, while not affecting the cutting operation directly, increase the susceptibility of the kerf faces to hardening; others slow down the cutting or, by forming refractory oxides, render it impossible. Much is dependent on whether a given steel contains an increased percentage of one impurity or it is supplemented by a high proportion of other impurities affecting the cutting operation.

Carbon. Up to 0.4 per cent carbon does not affect the cutting of steel. Over 0.4 or 0.5 per cent carbon impairs the efficiency of the cutting process, while over 1.0 or 1.2 per cent carbon renders it impossible.

As has been noted elsewhere, steels high in carbon are difficult to cut because their kindling temperature is higher and their melting point lower than of low-carbon steels. Another trouble due to the increased carbon content is probably the fact that much CO_2 is formed, contaminating the cutting oxygen, reducing the cutting speed, and impairing the quality of the kerf faces.

The flame cutting of steel is accompanied by the diffusion of carbon towards the surface of the kerf. As a result, the parent metal becomes carbon-impoverished at some distance from the kerf faces and carburised on and near them. This tendency becomes especially pronounced in steels high in carbon, and the metal adjacent to the kerf grows more susceptible to hardening.

The hardened structures forming near the kerf add to the hardness and brittleness of the metal in this area. Should the metal be cut under unfavourable conditions (low ambient temperature, complicated shapes, etc.), this may give rise to cracks.

Manganese. Up to 4 per cent manganese does not practically affect the cutting operation. More manganese handicaps the operation, while over 14 per cent manganese makes it unfeasible.

The combination of over 0.8 per cent manganese and over 0.3 per cent carbon makes the steel more susceptible to hardening, and the metal near the kerf grows hard and embrittled.

Silicon. When present in the quantities usual for steels, silicon does not affect the cutting operation markedly. An increased silicon content causes some trouble due to the formation of the refractory oxide SiO_2 which adds to the viscosity of cutting slag. The high percentage of silicon makes the cutting of steel impossible.

Phosphorus and sulphur. The quantities of these two impurities usually contained in steels do not affect the cutting operation.

Chromium. The effect of up to 4 or 5 per cent chromium on the cutting operation is that the slag grows more viscous and the kerf faces tend to harden. More chromium makes conventional flame cutting unfeasible because of much refractory Cr_2O_3 forming on the metal surface.

Nickel. Up to 7 per cent nickel does not hamper the cutting of steel. The cutting speed is somewhat reduced, though it remains satisfactory, as the nickel content rises to 34 per cent. According to some investigators, in high-nickel steels this element diffuses towards the kerf faces.

Molybdenum. When present in small amounts (under 0.25 per cent), molybdenum does not affect the cutting speed, but increases the hardenability of the metal and the hardness of the kerf faces.

Tungsten. When present in the usual amounts for alloy steels, this element does not hamper cutting. But, like molybdenum, it increases the hardenability of the metal and the hardness of the kerf faces. The oxidation is somewhat retarded by as much as about 10 per cent tungsten. Up to 20 per cent tungsten handi-

caps the cutting operation, while over 20 per cent renders it impossible.

Vanadium. The small amounts of vanadium usually found in steels do not affect the cutting operation. Its effect on the hardness of the kerf faces is similar to that of molybdenum, tungsten, titanium and other carbide-forming elements.

Copper. When it runs up to 0.7 per cent, copper does not appreciably affect the cutting operation.

Aluminium. When contained in negligible amounts (up to 0.5 per cent), aluminium has no effect on the cutting operation. When it runs over 10 per cent, steel is impossible to flame-cut.

6. The Effect of Oxygen Purity on Cutting

While in welding the purity of oxygen has little effect on the quality and efficiency of welding, it is of primary importance in flame cutting, affecting, as it does, cutting speed, gas consumption and cut quality. More specifically, when the purity of oxygen goes down, the oxidation of the metal is retarded, and more gas is consumed per unit length of the kerf, especially oxygen.

Investigators differ in their opinion about the effect of oxygen purity on oxygen consumption per unit length and cutting speed. It is customary to reckon that one per cent decrease in oxygen purity (within 99.5-97.5 per cent) causes an increase of 10 to 15 per cent in the cutting time per linear metre of steel plate and an increase of 25 to 35 per cent in the consumption of cutting oxygen. It should be noted, however, that other figures may be obtained, for the increase in oxygen consumption due to lower oxygen purity may also be aggravated by metal thickness (increasing for heavier gauges), by the natural retardation of metal oxidation, and by the rise in oxygen pressure usually resorted to by operators in order to be able to maintain the desired cutting speed.

If, when using oxygen of low purity, the operator fails to raise oxygen pressure in the torch, oxygen consumption will solely depend on how much the oxidation of the metal is retarded, or how much cutting speed is slowed down. If, on the other hand, the operator uses higher oxygen pressure, oxygen consumption will be governed by the slow-down in cutting speed and by the increase in gas pressure.

When oxygen purity is decreased by a limited amount, it is sound practice to slow down cutting speed a little, without readjusting gas pressure. The reason is that an increase in oxygen pressure often results in a wider cut and poorer kerf faces.

Table 40

Steel thick-ness, mm	Diameters, mm		Cut-ting oxygen pres-sure before torch, kg/cm²	Flow, m³/hr		Flash on bottom	Cutting speed, mm/min	Consumption; lit/lin. m	
	cutting oxygen orifice			acety-lene	oxy-gen			acety-lene	oxygen
	throat	outlet							
15	1.0	1.3	5.0	0.40	3.3	None	260-340	26-20	212-162
				0.45	3.4	Comes off easily	390-420	20-17	149-132
20	1.0	1.3	6.5	0.40	4.5	None	240-310	28-22	278-216
				0.45	4.1	Comes off easily	350-400	21-19	195-171
25	1.0	1.3	8	0.40	4.7	None	220-290	30-23	356-270
				0.45	4.8	Comes off easily	330-370	23-20	243-216
30	1.3	1.7	5.5	0.40	5.5	None	210-280	32-24	437-328
				0.45	5.6	Comes off easily	310-340	24-22	302-275
40	1.3	1.7	7	0.40	6.6	None	190-250	35-27	580-440
				0.45	6.7	Comes off easily	290-320	26-24	386-350
50	1.3	1.7	8.5	0.40	7.7	None	180-240	37-28	715-535
				0.45	7.8	Comes off easily	270-300	28-25	482-434
60	1.8	2.3	5	0.45	9.1	None	170-230	44-33	895-661
				0.50	9.2	Comes off easily	250-280	33-30	615-550
80	1.8	2.3	6.5	0.45	11.2	None	150-210	50-36	1,250-800
				0.50	11.3	Comes off easily	240-260	35-32	785-725
100	1.8	2.3	8	0.45	13.3	None	140-190	54-40	1,590-1,170
				0.50	13.4	Comes off easily	220-240	38-35	1,015-930

Notes: 1. The data have been tabulated for a torch held vertically.
2. Oxygen-to-acetylene ratio, $\beta_0 = 1.7$.
3. The inner cone is within 2 or 3 mm of the surface.

When oxygen purity considerably differs from the normal one oxygen pressure should be somewhat increased (thereby increasing its rate of flow) so as not to slow down the cutting speed.

Oxygen with a purity of below 97 per cent ought not to be used at all, for heavily contaminated oxygen makes deslagging a problem and impairs the appearance of the kerf faces.

Low oxygen purity increases the drag of the cut which is also dependent on metal thickness and cutting speed.

As oxygen purity increases, higher speeds may be safely employed without running any risk of impairing the quality of the cut. As has been shown by Hunnert and other investigators and field workers in East Germany, the highest efficiency of steel flame cutting and the highest quality of the cut (with the least flash) can be obtained with the maximum attainable purity of oxygen, or 99.8 per cent.

This has been borne out by researchers at the Autogenous Welding Research Institute who have been able to cut low-carbon steel from 3 to 100 mm thick without any flash with oxygen 99.5-99.8 per cent pure.

Yet, they have been able to show that in steel sections over 12 mm thick, cutting with little, if any, flash and easily removable slag is possible with an oxygen purity of 98.5 per cent, provided the cutting variables are matched properly (Table 40).

7. The Effect of Cutting on the Structure and Properties of Steel

The heat generation and chemical reactions accompanying the cutting operation may change the structure, analysis and mechanical properties of the metal.

(1) Structural Changes. Some of the heat supplied by the oxidation of the metal and by the preheating flame is absorbed by the cut edges. The edges are raised to melting point and then cool rapidly, as the heat is transferred to the adjacent cooler metal by thermal conductivity and to the surrounding atmosphere by radiation.

The structural changes due to the heat effect mainly depend on the thickness and analysis of the metal, being more pronounced in heavier gauges. The increased content of carbon and of some hardening elements acts in the same direction.

Plain low-carbon steel is not practically hardened by cutting. The effect of the cutting heat on this steel is limited to grain growth in the heat-affected zone and the appearance of unbalanced sorbite in addition to pearlite. As a rule, the sorbite is little developed and brings about insignificant changes in the steel

grains. This is because during the cutting operation the cut edges remain at above the upper critical point, Ac_3, for too short a time (as the metal cools rapidly), for austenitic homogenisation to occur, and the metal structure remains heterogeneous. Besides, the high rate of heating, characteristic of the cutting operation, shifts the transformation interval ($Ac_1 - Ac_3$) towards the region of higher temperatures, and only partial recrystallisation takes place in the fairly wide heat-affected zone.

The presence of sorbite in the heat-affected zone does not impair the quality of the metal. In fact, its mechanical properties, especially its ultimate tensile strength, are improved without reducing its ductility.

In the case of steels higher in carbon or containing alloying elements, structural changes cover more of the adjacent metal, and in addition to pearlite and sorbite, the structure may include troostite and even martensite which adds to the hardness and brittleness of the metal.

The presence of martensite impairs the machinability of the flame-cut edges and raises the susceptibility of the metal to cooling cracks.

Cooling cracks are most likely to develop in heavy gauges of air-hardening steel, especially when cutting complex shapes or closed outlines. Most often cracks appear where locked-up stresses concentrate (at right and acute angles, etc.).

The cutting of hardened steels without annealing or normalising prior to cutting, increases the danger of cracking in the cut edges.

The ways and means for controlling the hardenability of the metal due to cutting are discussed in the chapter dealing with the cutting procedures for structural steels.

(2) Changes in Analysis. Flame cutting affects the analysis of the metal both in the cut edges and in the heat-affected zone. In the edges, these changes may be brought about by the direct action of the flame and cutting oxygen on the metal and by the diffusion of some elements towards the kerf. Changes in the analysis of the more remote mass of metal may only be caused by diffusion.
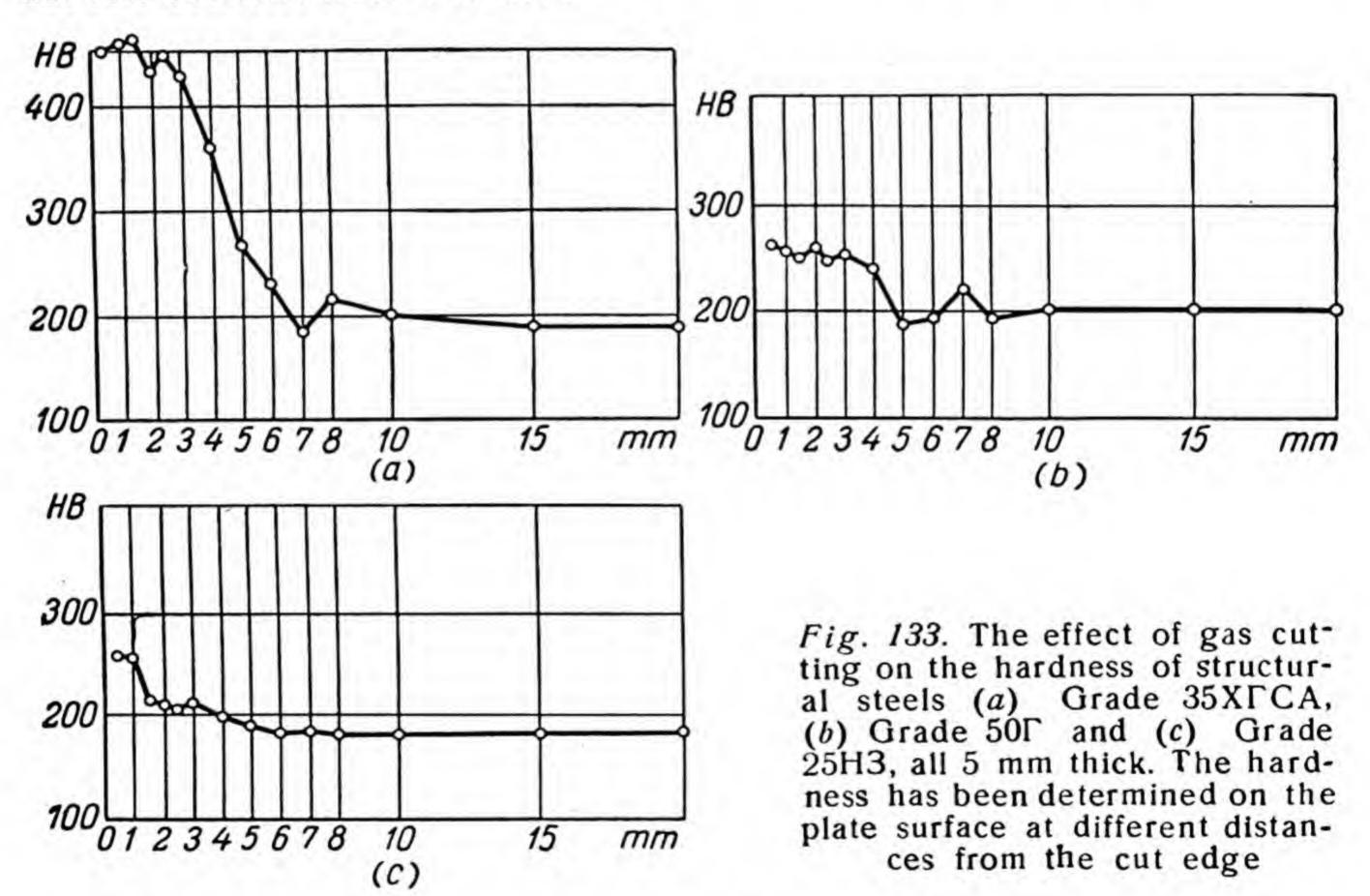
In low- and medium-carbon steels, a decarburised layer is formed on the kerf faces, several tenths of a millimetre thick. This layer is produced by the burning of carbon and other constituents with the oxygen jet. The decarburised layer is followed by a layer in which the carbon content is usually higher than in the original metal. Farther from the kerf, the carbon content progressively decreases until within some distance of the kerf

which depends on metal thickness and cutting conditions, it drops to below the original carbon content. Past this layer, there is as much carbon present as in the original steel.

Nickel, too, moves towards the kerf. The other elements burn to different degrees, depending on their affinity for oxygen, and they do so only on the surface of the edges, where the oxygen jet comes in direct contact with the metal.

(3) Changes in Hardness. In the case of low-carbon steel containing not over 0.2 per cent carbon, the cutting operation affects the hardness of the metal little and solely due to the increased carbon content near the kerf.

Fig. 133. The effect of gas cutting on the hardness of structural steels (*a*) Grade 35ХГСА, (*b*) Grade 50Г and (*c*) Grade 25Н3, all 5 mm thick. The hardness has been determined on the plate surface at different distances from the cut edge

With carbon running more than 0.2 per cent, the cutting operation may raise the hardness of the steel near the kerf.

Structural steels containing alloying elements and an increased amount of carbon show greater changes in hardness both in the cut edges and in the heat-affected zone. Due to the greater hardenability of these steels, this increase in hardness reaches deeper into the metal (Fig. 133).

CHAPTER XXI

THE CUTTING OXYGEN JET AND CUTTING TIPS

1. Comparison of Existing Tips

To be efficient, the cutting operation requires that exact quantities of oxygen be supplied to the torch.

With too little oxygen, the iron in the line of cut fails to burn completely and the resultant oxides are not removed efficiently. An excess of oxygen cools the metal being cut.

The quantity of oxygen passing through the tip depends on the velocity of the oxygen jet which in turn depends on the construction of the tip, its throat and the pressure of the oxygen.
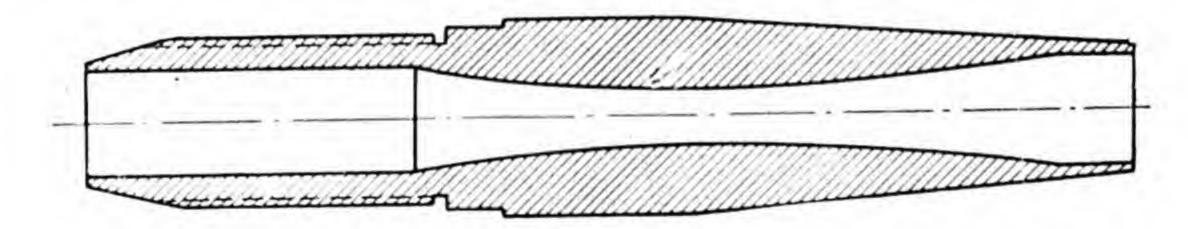
Fig. 134. A cutting oxygen tip with an expanding orifice

The oxygen jet issuing from the cutting orifice should oxidise the metal across its entire thickness continually, and the cutting torch should be advanced along the line of cut in proportion to the rate of oxidation across the thickness of the workpiece.

The rate of oxidation is greatly affected by the velocity at which the oxygen jet issues from central orifice.

For the oxides forming in the cut to be removed as soon as they are produced, it is desirable that the velocity of the oxygen jet be as high as practicable.

Furthermore, if the oxygen jet is to produce a cut of constant width across the entire thickness of the metal and to have the least possible drag (see Sec. 2 of this chapter), the cutting oxygen jet should retain its straight shape for as great length as possible.

The maximum velocity of the cutting oxygen jet, which exceeds that of sound, and the vital straight shape of the jet can both be

obtained by means of tips such as shown in Fig. 134. When made sufficiently long, such tips provide for the complete expansion of the oxygen across the outlet orifice to the pressure of the surrounding atmosphere and, consequently, for the complete conversion of the potential energy of the gas to kinetic.
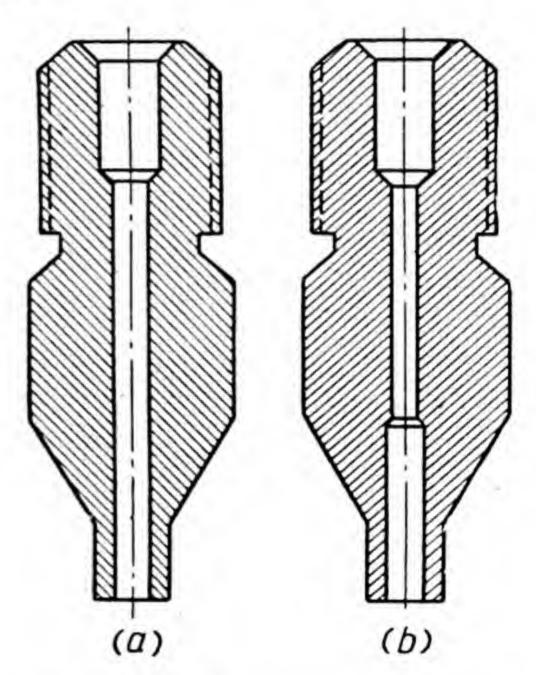
Under the conditions of the adiabatic flow of gases, the maximum rate of oxygen flow through a tip will be at the critical pressure ratio, or

$$\beta = \frac{p_{sur}}{p_1} = 0.528. \qquad (XXI.1)$$

Under the conditions of flame cutting, the pressure of the surrounding atmosphere, p_{sur} is 1 atm (abs). Therefore, the critical pressure ratio, when supersonic speed is reached, is attained at an oxygen pressure $p_1 = 0.893$ atm (gauge).

Fig. 135. Cutting oxygen tips:
a — with straight-sided orifice; *b* — with stepped straight-sided orifice

(a)   (b)

In tips with straight-sided orifices (Fig. 135 *a*), when the inlet oxygen pressure is over 0.893 atm (gauge), the gas fails to expand fully at the outlet, and its pressure exceeds that of the surrounding atmosphere. Therefore, the pressure ratio in such tips is below critical, i. e., less than 0.528. Furthermore, the expansion of the oxygen takes place outside the tip, and the oxygen jet is more spread than in the case of tips with expanding orifices (at the same working oxygen pressures).

Light-interference studies of oxygen jets have shown that tips with straight-sided or stepped straight-sided orifices (Fig. 135 *b*) produce near cylindrical jets of oxygen only at relatively low oxygen pressures (as measured in the oxygen regulator), usually not exceeding 5 or 6 atm (gauge). At high

pressures the cutting oxygen jet diverges into a cone. This is especially true of tips with straight orifices. Tips with divergent orifices produce cylindrical jets at pressures twice as high.

Yet, when cutting light-gauge steel, good cuts may be obtained with a working oxygen pressure of over 6 atm (gauge), using
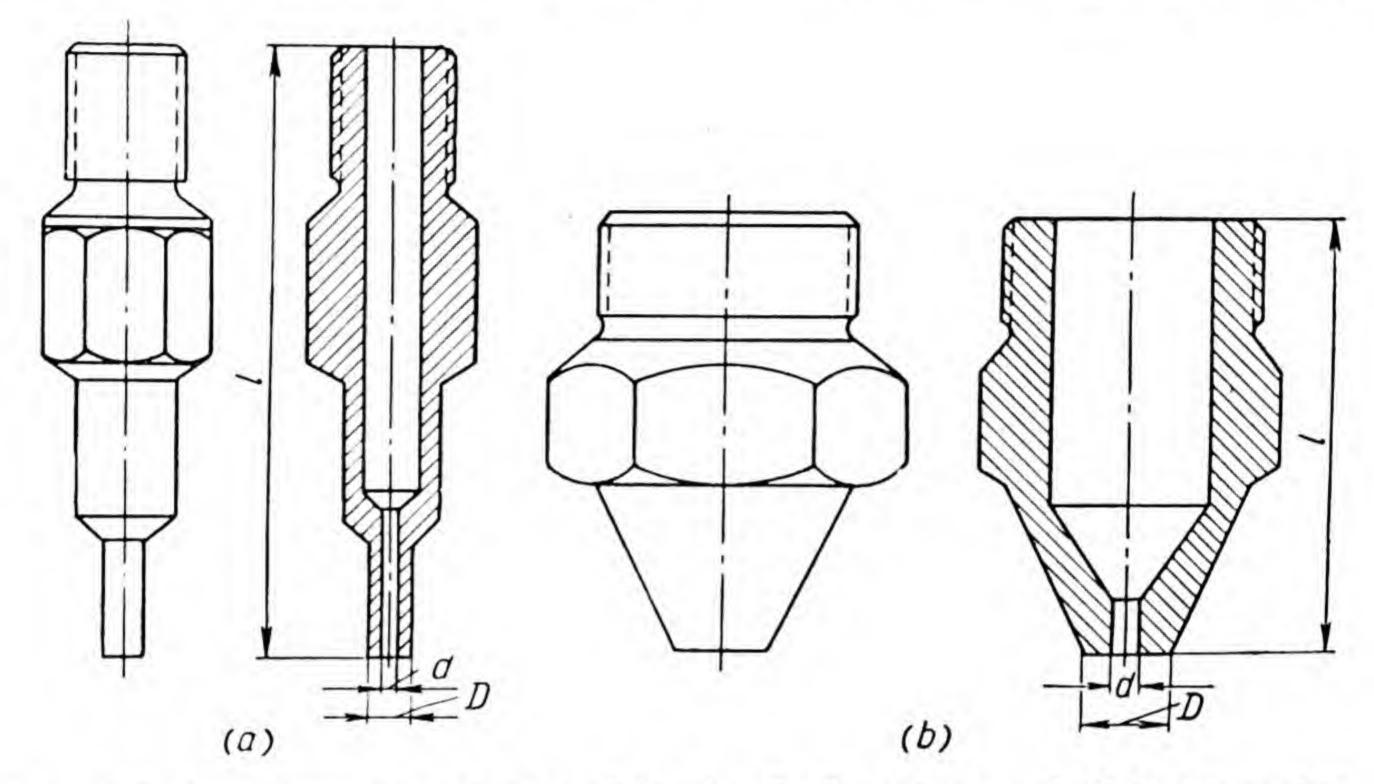

Fig. 136. Two-piece tips developed by A. G. Semyonov and N. S. Eismontas:
a — inner nozzle; *b* — outer nozzle

Inner Nozzles					*Outer Nozzles*			
Size	D	d	l		Size	D	d	l
0	1.8	0.3	42		0	5	2.2	30
1	2.5	0.5	42		1	6	2.7	30
2	2.5	0.75	42		2	6	2.9	30
3	2.5	1.0	42		3	6	3.0	30

tips with straight-sided and, the more so, with stepped straight-sided orifices. In this connection, mention should be made of the composite tips developed for commercial torches by engineer Semyonov and operator Eismontas. A composite tip consists of an internal nozzle which has an elongated cutting oxygen orifice (Fig. 136 *a*), and an external nozzle with an elongated and converging outlet orifice (Fig. 136 *b*). This tip imparts greater velocity to both the cutting oxygen jet and the oxy-gas mixture of the preheating flame, with the result that the heat is more con-

centrated, the heating rate is increased, as are the oxidation of the metal and the removal of the oxides. As the inventors of this tip believe, an important part is played by the increased annular space between the internal and external nozzles, where the oxy-gas mixture of the preheating flame travels, as it makes the flame more stable and its inner cone longer.

The two inventors have developed two sizes of composite tips (differing in the orifice diameters) which have proved a success at the Klaipeda Shipbuilding Yards. Both sizes are claimed to offer the following advantages:

(1) an increase of 20 to 100 per cent in cutting speed for steel up to 100 mm thick with the Size No. 1 tip and an increase of 55 to 230 per cent in cutting speed for steel up to 40 mm thick with the Size No. 2 tip;

(2) a saving of as much as 50 per cent in acetylene and oxygen, as compared with standard tips;
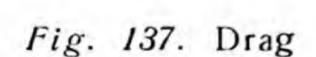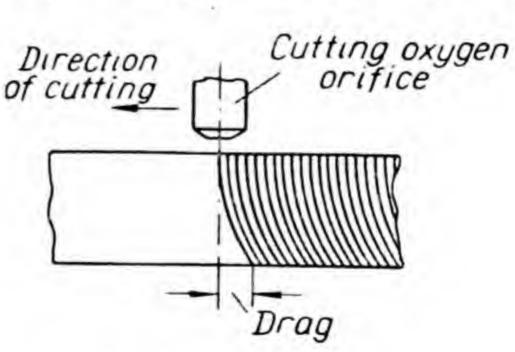
(3) a reduction of 50 per cent in kerf width;

(4) no flash on the cut edges;

(5) reduced buckling due to cutting.

2. The Effect of Oxygen Velocity and Jet Shape on the Drag, Quality and Rate of Cutting

In cutting, especially where heavy sections are involved, the metal at the top of the cut is oxidised quicker than at its bottom. The contributing causes are (1) contamination of the cutting

Fig. 137. Drag

oxygen with products of oxidation as it passes through the ma-terial; (2) no preheating of the metal at the bottom; (3) reduction in the kinetic energy of the oxygen jet; and (4) increase in the diameter of the jet away from the orifice, so that the bottom metal is oxidised over a greater width and more slowly.

As a result, the oxygen jet may curve backward in a direc-tion opposite to its travel (Fig. 137). When this occurs, the amount by which the oxygen jet falls behind the perpendicular in passing through the material is known as the "drag" of the

cut. Faint markings called "drag lines" left on the face of the kerf enable this to be measured.

The drag is inevitable in cutting both heavy and light gauges, though its magnitude may be different.

The drag is of minor importance in making straight-line or gently curved cuts, especially when the face of the cut is not to meet any special requirements and the drag does not interfere with the cutting operation. In shape cutting of heavy sections, the drag may be objectionable, as the bottom contour of the cut tends to come out differently from the top, with the edges not square. As a precaution against this condition, cuts should be made with short drag, which requires a lower cutting speed.

In the case of straight-line cuts made at normal cutting speed with tips having stepped straight-sided orifices, the drag is usually as follows:

Metal thickness, mm	Drag, mm
5-25	1-5
25-50	5-8
50-100	8-12
100-200	12-15

The drag mainly depends on the tip used, i. e., the velocity of the issuing jet of oxygen and its shape. To a lesser degree, it depends on cutting speed and metal thickness, increasing as the latter two factors are increased.

In heavy steel sections, high-quality cuts, i. e., cuts with small or no drag at all, constant kerf width and smooth cut faces free from grooves, flutes or ragged spots, can be easily obtained with well-designed tips in which the orifice diverges at an angle of 10 or 11 degrees. The cutting oxygen jet they produce is nearly cylindrical in shape. Higher rates of cutting are likewise obtained with flared tips, as compared with the straight-sided type used at regular oxygen pressures and usual oxygen line diameters.

To sum up,

(1) Straight-sided orifice tips of efficient design, such as the Semyonov type, can make high-quality cuts at a reasonably high rate of cutting in steel up to 100 mm thick, using ordinary oxygen pressures (not over 6 atm gauge).

(2) Stepped straight-sided tips, as compared with straight-sided ones, allow the gas to expand more at the orifice outlet and more of the potential energy of the gas to be converted to kinetic. In other words, they produce higher jet velocities and less divergent jets. When used in conjunction with ordinary oxygen pressures (3 to 14 atm, gauge), these tips may make qua-

lity cuts at a reasonably high rate of cutting in steel 4 or 5 to 300 mm thick.

(3) Well-designed expanding tips (with an angle of divergence of 10 or 11 degrees) of sufficient length, when used in conjunction with oxygen lines of larger diameter and a reduced oxygen working pressure, ensure the highest velocity of the oxygen jet which remains fundamentally cylindrical. As a result, high-quality cuts and high rates of cutting can be obtained on extremely heavy sections of steel (up to 1.5 or 2 metres thick).

(4) No appreciable divergence of the oxygen jet occurs, however, in cutting steel up to 600 or 700 mm thick with stepped straight-sided and even straight-sided tips if used in conjunction with oxygen lines of increased diameter and a reduced pressure of cutting oxygen (6 to 8 atm, gauge, at the regulator and not over 2.5 atm, gauge, at the orifice). * Therefore, expanding tips cease to be a must.

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* For a detailed discussion of low-pressure oxygen cutting see Sec. 2, Chapter XXIII.

CHAPTER XXII

FLAME-CUTTING EQUIPMENT

1. Manual Cutting Torches

(1) **Classification.** Manual cutting torches may be classed on the basis of the fuel gas they burn and the purpose they are intended to serve.

On the basis of the fuel gas burned they are classed into (a) oxy-acetylene torches; (b) oxy-gas torches (using hydrogen, methane, natural gas, oil-refinery gas, propane-butane or propane-propylene mixtures, etc.) and (c) oxy-benz torches (burning kerosene, petrol, benzene or other liquid hydrocarbons).

By the purpose served they are grouped into (a) universal, capable of making straight-line, circular and shape cuts in steel up to 300 mm thick and (b) special-purpose, for cutting heavy steel sections, underwater cutting, hole cutting, rivet cutting, etc.

(2) **Requirements.** Cutting torches are similar in basic design to welding torches. The difference lies in provision of an additional tube for supplying the cutting oxygen, complete with a valve to open and shut off the oxygen supply, as well as in the construction of the head and tip.

Like their welding counterparts, cutting torches may be of the injector type. The injectors for cutting torches may operate on both low and medium fuel-gas pressures.

An efficient cutting torch should meet the same requirements as a good welding torch (see Chapter IX). There are, however, some additional requirements to them, namely:

(a) the possibility of cutting in any direction;

(b) the proper ratio of the preheating to cutting oxygen orifice areas (otherwise, either the preheating flame or the cutting oxygen jet will be unfit for cutting a given metal thickness);

(c) concentricity in the relative position of the outer and inner nozzles of a composite tip;

(d) the possibility of adjusting the flame with the cutting oxygen turned on;

(e) high surface finish inside the tip, especially in the cutting oxygen orifice; freedom from scratches, burrs or other roughness;

(f) leak-tight joints throughout;

(g) the possibility of using interchangeable tips suitable for various metal thicknesses;

(h) sufficient length to protect the operator; and also provision of a protective peak on torches intended for heavy gauges;

(i) provision of guide wheels on which to advance the torch at a constant rate and at a constant distance from the metal surface.

(3) Construction. Fig. 138 shows diagrammatically an injector-type oxy-acetylene cutting torch. It consists of a shank and a head. The shank embraces a handle *1*, hose connections *2* and *3* (one for the acetylene hose and the other for the oxygen hose),
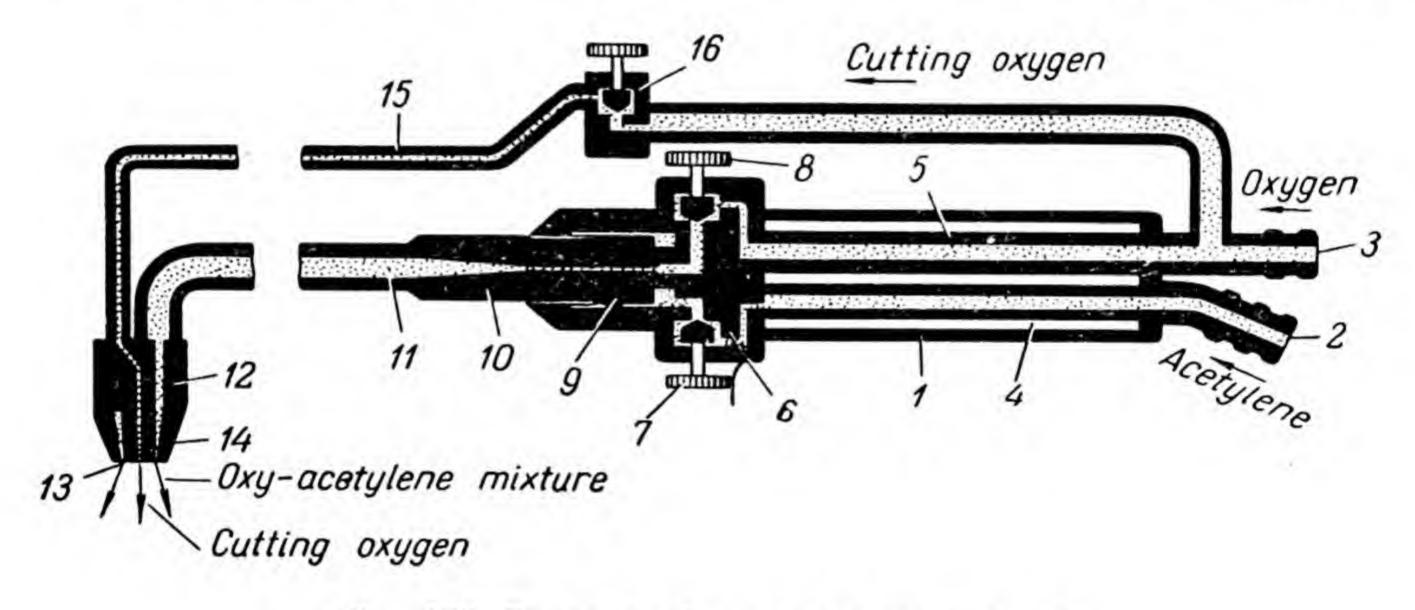

Fig. 138. Diagram of an injector-type torch

an acetylene *4* and an oxygen *5* tubes, a valve body *6* incorporating an acetylene valve *7* and an oxygen valve *8*. The head is made up of an injector *9*, a mixer *10*, a preheating flame tube *11*, a two-piece tip *12* consisting of an inner nozzle *13* and an outer nozzle *14*, a cutting oxygen tube *15*, and a cutting oxygen valve *16*.

The acetylene is supplied under a low gauge pressure through the connection *2* whence it enters the tube *4*, flows through the valve *7* and into the side passages in the injector *9*. The oxygen is under a much higher pressure; it is admitted through the connection *3*. Passing through the valve *8* and the injector *9* at a high velocity, it creates the sucking action in the mixing cham-

ber *10* and the acetylene is drawn into it to be thoroughly mixed with the oxygen.

The gas mixture leaves the mixing chamber by the preheating flame tube *11*, passes through the head *12* and issues from the annular orifice between the inner and outer nozzles *13* and *14*; in burning, the mixture produces the preheating flame.

Opening the valve *16* admits oxygen into the tube *15* which supplies it to the tip *12* where it issues from the inner nozzle *13* to produce the cutting oxygen jet which oxidises the metal in the line of cut and blows the resultant oxides out of the kerf.

The above features are embodied in Type PP torches, in detachable РГС and РГМ torches, and in Type P3P torches burning gases other than acetylene.

(4) Oxy-gas Torches. This group is examplified by the Type P3P-55. It is intended for the manual cutting of low-carbon steel and burns natural gas, methane, city gas, coke-oven gas, propane, or propane-butane mixture in the preheating flame.

A typical torch in this group usually comes complete with assorted internal and external nozzles, a two-roller truck to maintain the distance between the tip and the metal constant, and a radial bar for circular cuts.

(5) Oxy-benz Torches. This type of torch burns kerosene or petrol vapours in the preheating flame.

In fact, the liquid fuel may either be evaporated or atomised before it is burned.

The evaporation principle is employed in the Type K-51 cutter developed by the Autogenous Welding Research Institute in 1951. It consists of a cylinder holding 5 litres of liquid fuel, a suitably designed torch and hoses. Fig. 139 shows a general view of the K-51 cutter.

The fuel cylinder has a spherical head which carries an air pump fitted with a nonreturn valve, a tube, a shut-off valve receiving a kerosene-resistant hose, and a pressure gauge.

In priming the torch, the air pump creates a pressure of 0.5 to 2 atm (gauge) above the fuel in the cylinder. This pressure forces the fuel down the hose and into the torch. Fig. 140 shows the construction of the torch diagrammatically.

The oxygen is supplied through the connection *1*, valve *2*, tube *3* and injector *4* into the mixing chamber *5* where it is mixed with the fuel vapours coming from the asbestos-packed evaporator *8* where it comes from the hose by way of the connection *6* and the tube *7*. The evaporator is heated by the flame burning at an auxiliary nozzle *9*.

The proportion of the fuel is adjusted by the handwheel *10* coupled with the tube *3*. Clockwise rotation of the handwheel
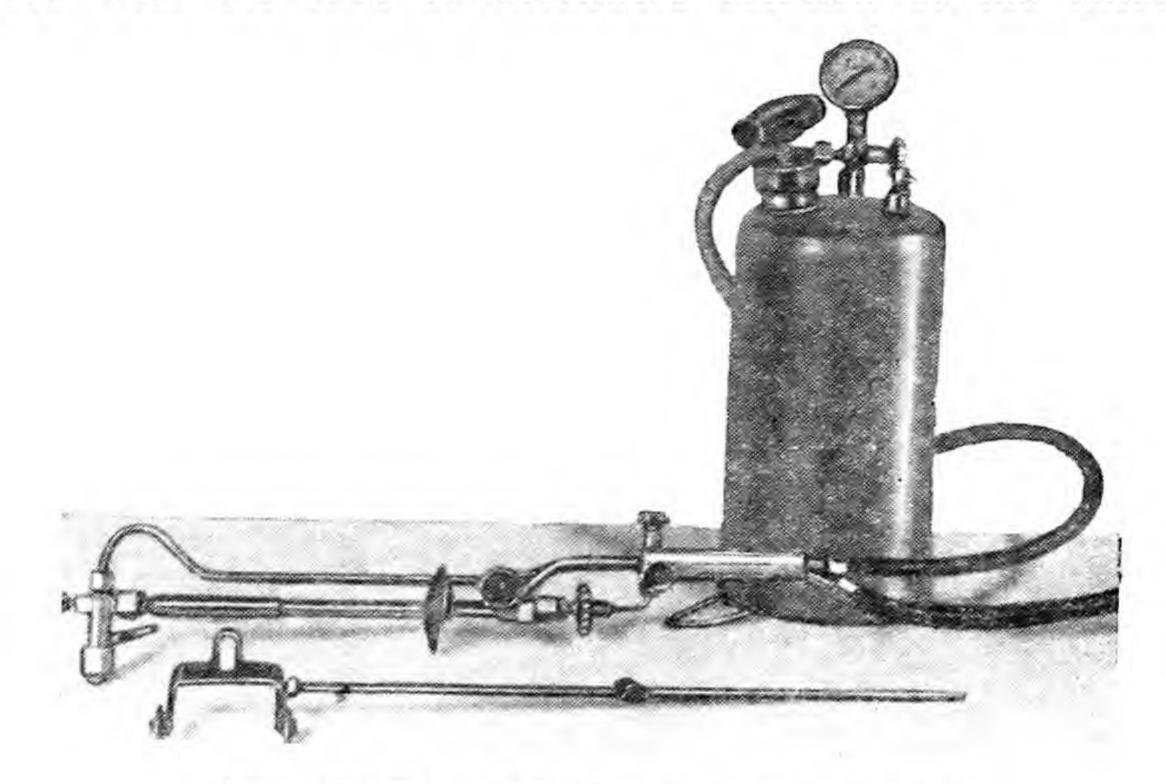

Fig. 139. Type K-51 oxy-benz cutting outfit

drives the tube into the torch body, and the outlet orifice for fuel vapours is reduced in area. Counterclockwise rotation of the
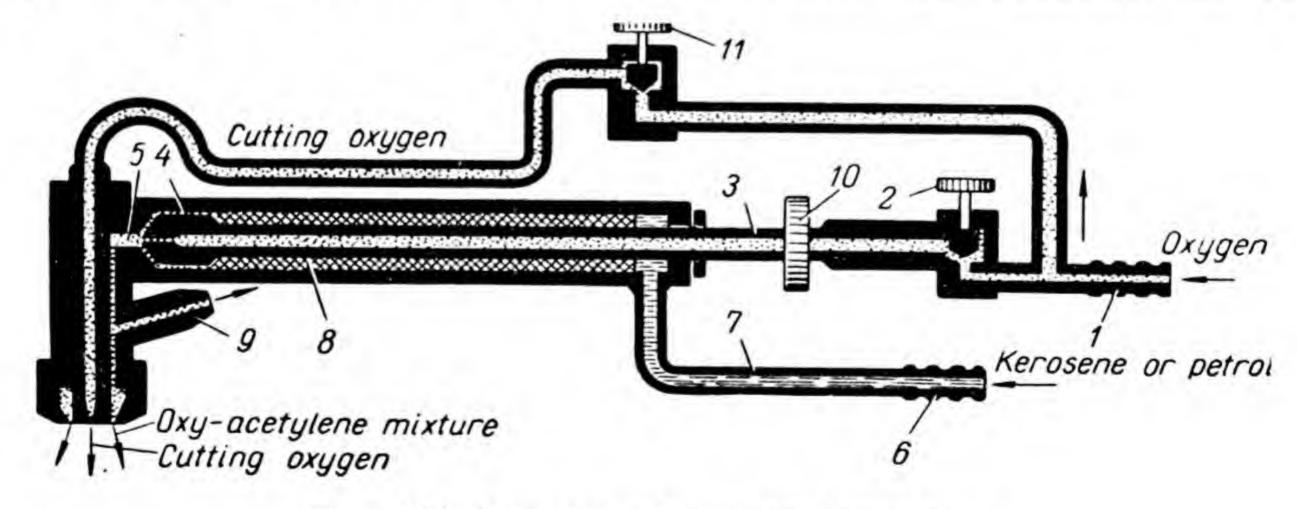

Fig. 140. Diagram of an oxy-benz torch

handwheel increases the area of the outlet orifice, and the flame output is increased.

The proportion of the oxygen for the preheating flame is adjusted by the valve *2*, while the cutting oxygen is admitted and shut off by the valve *11*.

Table 41 gives the basic technical data on the oxy-benz torch.
As compared with earlier makes, the Type K-51 incorporates the following improvements: (a) a handle; (b) a shut-off valve on the kerosene line; (c) increased orifices for cutting oxygen and a straight tube supplying the cutting oxygen from the shank

Table 41

Technical Data on the K-51 Oxy-benz Torch

Characteristic	Metal thickness, mm			
	up to 20	20-50	50-100	100-200
Size of internal nozzle	1	2	3	4
Oxygen pressure, kg/cm²	4-5	5-7	7-9	9-11
Fuel pressure in cylinder, kg/cm²	1.5-3.0	1.5-3.0	1.5-3.0	1.5-3.0
Oxygen flow rate, m³/hr	5.4-7.6	7.6-9.8	9.8-20.2	20.2-32.6
Kerosene flow rate, kg/hr	0.7-0.8	0.8-0.9	0.9-1.1	1.1-1.3
Oxygen consumption, lit/m	134-423	423-1,090	1,090-3,360	3,360-7,230
Kerosene consumption, g/m	25-53	53-100	100-180	180-290
Cutting speed, mm/min	500-300	300-150	150-100	100-75

Note: The table applies to vertical straight-line cuts made with oxygen at least 99 per cent pure and in steel containing not over 0.3 per cent carbon.

to the head; (d) improved seals in the evaporator and the head; (e) a seal preventing kerosene from finding its way into the air pump and from running out should the valve become leaky.

In using an oxy-benz torch, it is important to see that the pressure in the fuel cylinder never exceeds the oxygen pressure as maintained by the regulator. Otherwise, a flash-back may set fire to the kerosene hose.

Instead of kerosene, petrol may be used as the fuel gas, which requires that the nozzle heating the evaporator have a smaller orifice (table 42 and Fig. 141).

Sometimes, when cutting in cold weather, a small quantity of petrol may be added to kerosene in order to raise flame output and to make up for the heat loss due to low ambient temperature.

The principle of fuel atomisation is employed in the PKP-3-57 cutter developed by the Autogenous Welding Research Institute

in 1957, in which the liquid fuel is atomised by oxygen and is then evaporated in the tip. The torch (Fig. 142) consists of a head holding an internal and an external nozzle, tubes, shank, filter and nonreturn valve.

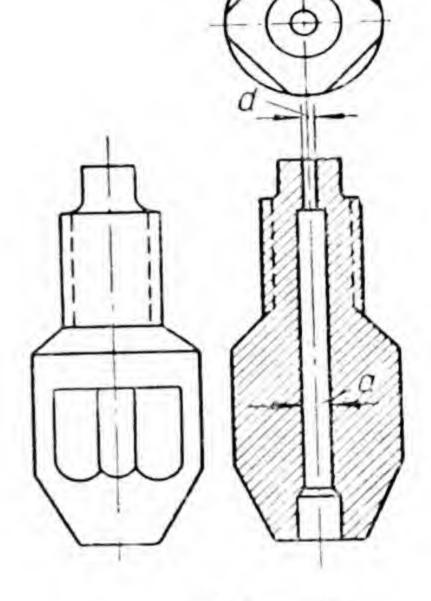*Fig. 141.* A tip to heat the kerosene evaporator in an oxy-benz cutter

Table 42

Evaporator Nozzle Diameter

Fuel	Diameter, mm	
	a	b
Kerosene	1.2	0.9
Petrol	1.0	0.45

The fuel from the cylinder is supplied by a kerosene-resistant hose to the filter whence it flows through an adjusting valve, a tube and into the atomiser in the torch head.

Overcoming the nonreturn valve spring, the oxygen enters the torch shank where it is divided into two streams, one reaching the adjusting valve and the other the shut-off valve.

Passing through the adjusting valve, the tube and the atomiser the oxygen leaves the latter by the side openings to break up the jet of liquid fuel. The mixture of atomised fuel and oxygen thus produced passes through the annular space between the inner and outer nozzles and vapourises by coming in contact with the hot wall of the outer nozzle (when the cutting operation proceeds with a steady rate). The vapours issue from the preheating flame tip.
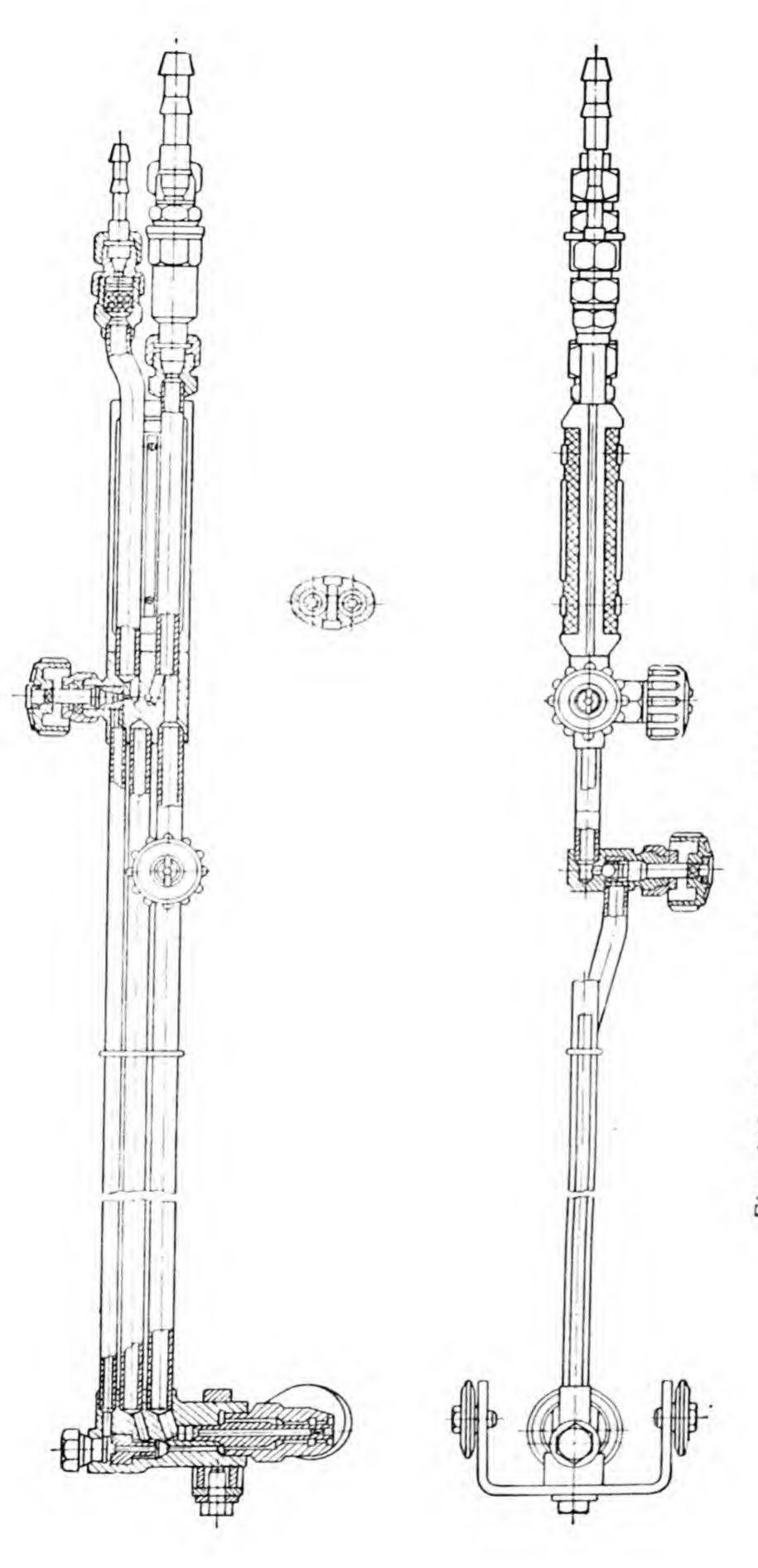
Fig. 142. The torch of the Type PKP-3-57 oxy-benz cutting outfit

The cutting oxygen passes through the shut-off valve, the torch head, the inner nozzle and into the kerf.

The valves in the torch are sealed with fluorinated plastic. PKP-3-57 oxy-benz torches come complete with one atomiser, two outer nozzles (see Table 43), and a Type БГ-1 liquid-fuel tank.

To start the preheating flame, a half-turn should be given first to the fuel valve and then to the preheating oxygen valve.

Table 43

Performance Data on PKP-3-57 Torch

Metal thickness, mm	5-18	18-30	30-50	50-100	100-200
Nozzle size outer . . .	1	1	1	2	2
inner . . .	1	2	3	4	5
Pressure, kg/cm^2:					
kerosene	1.5-3	1.5-3	1.5-3	1.5-3	1.5-3
oxygen	4.6-8	6.5-8.5	6.5-8.5	6.5-8.5	6.5-8.5
Kerosene consumption, kg/hr	0.5-0.7	0.7-08	0.8-0.9	0.9-1.0	1-1.2
Oxygen consumption, m^3/hr	3-4	5-7	8-9	11-15	18-24
Cutting speed with short drag, mm/min	490-320	320-270	270-230	230-180	180-150

(6) Special-purpose Torches. This group embraces torches using detachable tips intended for hole cutting, smoke tube cutting, and rivet cutting (Table 44).

Table 44

Soviet-made Special-purpose Torches

Torch	Pressure, kg/cm^2		Flow, m^3/hr		Length of detachable head, mm	Weight of detachable head, kg
	oxygen	acety-lene	oxygen	acetylene		
PAO-01-55 (for hole cutting)	3-8	—	2.5-7.5	0.35-0.9	290	0.6
PAT-01-55 (for smoke tube cutting) . . .	3-5	—	2.5-4	0.35-0.5	290	0.5
PA3-01-55 (for rivet cutting)	3-8	—	4-7	0.5-0.7	370	0.6

2. Machine Torches

Classification. As distinct from manual cutting torches, machine torches use one-piece tips with six concentrically arranged preheating flames and a central cutting oxygen orifice.

Machine torches are classed into (a) those intended for general-purpose cutting machines and (b) those intended for special-purpose cutting machines. Those in (a) are made up of standard parts.

They may be of the oxy-acetylene and of the oxy-benz type. Oxy-acetylene machine torches are available in several commercial sizes (Table 45).

<div align="right">*Table 45*</div>

Torch	PM-1	PM-2	PM-3		PM3-Г
Description	Single-valve, 220 mm long	Double-valve, 220 mm long	Triple-valve, 160 mm long	Triple-valve, 280 mm long	Triple-valve, with horizontal connections, 280 mm long
Purpose	To ПЛ-2 cutter	To ПП-2 cutter and АЛП and АСП machines	To ПП-1 cutter	To self-propelled СГ-2 cutting head	To TP machine

3. Cutting Machines

The cutting machines commercially available in the Soviet Union are classed into general- and special-purpose types.

General-purpose machines are subdivided into portable МРТ tractor-mounted machines and stationary МРЛ, МРК, МРШ and МРП machines. Portable МРТ machines can be easily moved about and set up at any place in the shop, either directly on the plate to be cut or on a portable guide track. They are mainly used to make straight-line and circular cuts. In the latter case, they employ radial bars. Light-weight models of МРТ machines can have a crank with which to guide them along the line of cut. Some portable machines are motor-driven and fitted with tracing devices which automatically follow a template of the required shape mounted directly on the work.

Stationary cutting machines are for the most part automatic units equipped with tracing devices and capable of both straight-line and shape cutting.

Special-purpose machines are available in a wide variety of makes and designs. Some of them, intended for straight-line cuts, are semiautomatic units, with the torch guided manually round the profile. Others are complex stationary machine tools fitted with tracing devices following templates of very involved design automatically.

The proper selection of a cutting machine for the job on hand cannot be emphasised too much, if any waste of power and machine time is to be avoided. Straight-line and circular cuts may advantageously be made by MPT portable machines. When, on the other hand, a repetitive job of complex shape should be cut by an in-line cutting machine, the latter should preferably be a motor-driven stationary unit with motor-driven guide rollers. In general, the factors to be taken into account in choosing a particular type of machine are the availability of metal-cutting equipment, the scope of work on hand, and the advisability of investing money in new equipment not locally available.

1) Portable MPT Tractor-mounted Machines. MPT machines can be driven either electrically, or pneumatically (by oxygen or air) or mechanically (by a spring motor).

Electric-driven machines are most common and include both MPT models and the Type ПП of a later vintage. Developed by the Autogenous Welding Research Institute, the ПП machine is a light-weight portable unit of good stability and with a body placed rather high.

The small weight and size make the machine highly manoeuvrable and useful in general engineering, shipbuilding and construction. The machine is driven by a Type СЛ-322, 22-W, 110/127 V a-c motor. The model ПП-1 has one torch, and the model ПП-2 two torches (Fig. 143).

The ПП-1 machine can make straight-line cuts, circles with a diameter of 540 mm or more, and straight and curved profiles, provided they are gently curved.

Using tip sizes Nos 6.5 and 7, the machine can, respectively, sever steel up to 250 mm thick, cut low-carbon steel 5 to 30 mm thick at a high rate and flux-cut stainless steels 5 to 100 mm thick.

The machine may be guided either by a straight or a curved template (straight or curved angles measuring 45 × 45 × 5, 40×40×5 or 35×35×5 mm), by a semirigid guide track, a radius rod or by hand.

The speeds available in the ПП-1 machine are 80-380 mm/min and 320-1,500 mm/min, adjustable by means of change gears.
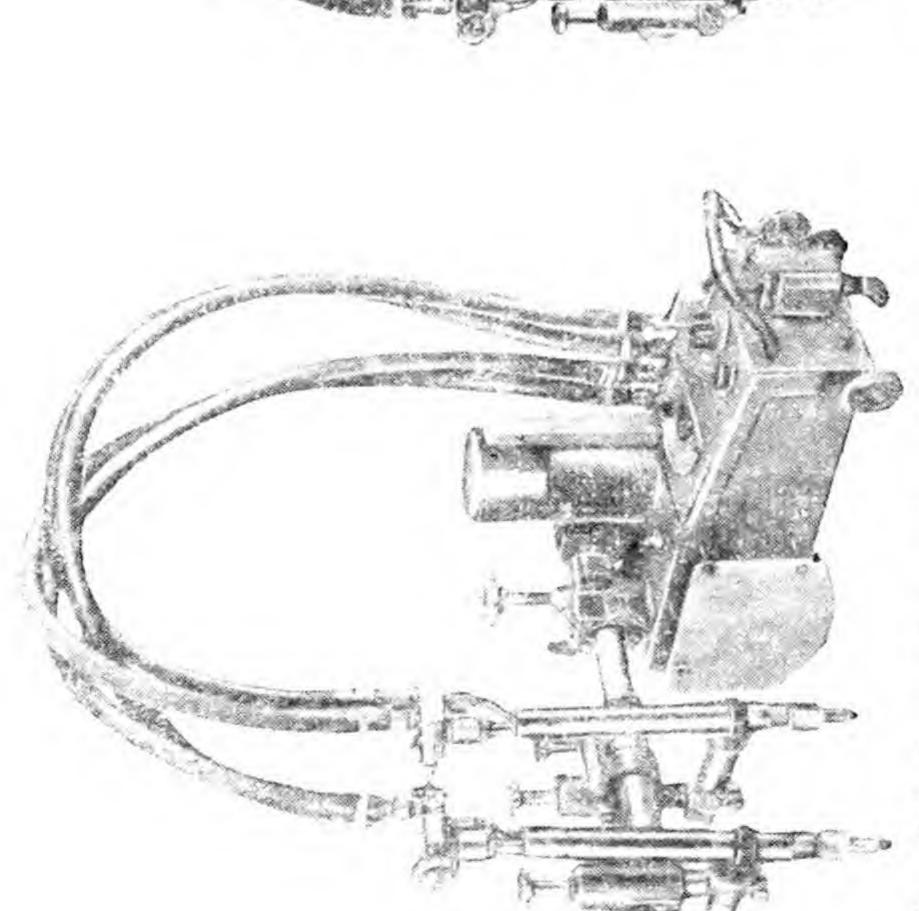
Fig. 144. Type ППВ-2 air-operated tractor-mounted cutting machine
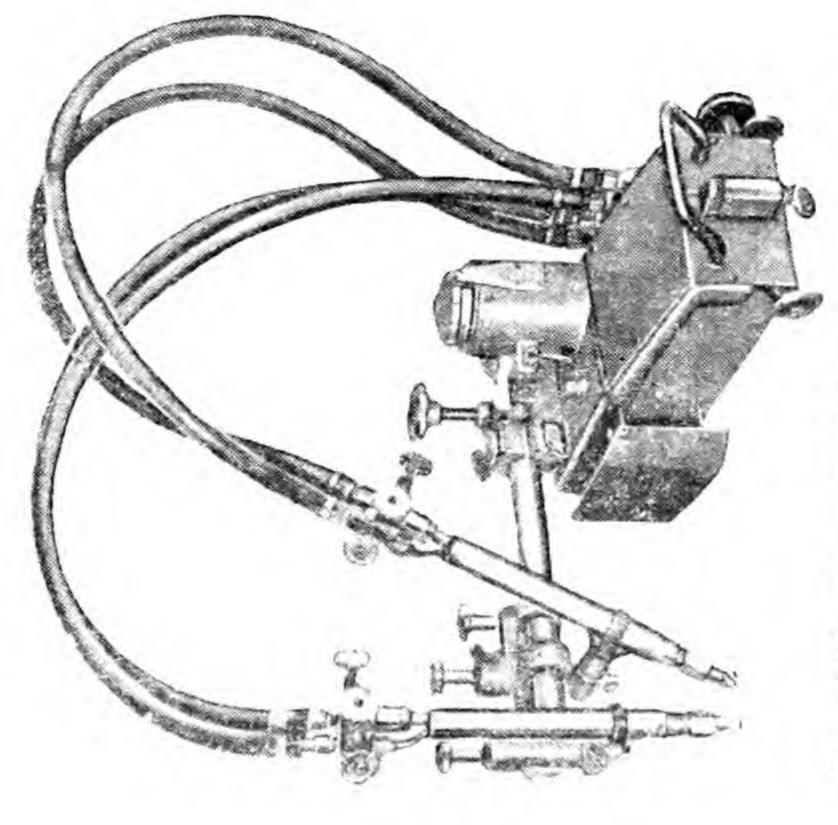

Fig. 143. Type ПП-2 electric-driven tractor-mounted cutting machine

The ПП-2 two-torch machine, in addition to the operations performed by the ПП-1, can make straight-line, circular and bevel cuts, with the two torches burning simultaneously.

The Autogenous Welding Research Institute has also developed the ППВ machine, which is basically a ПП machine with the electric motor replaced by a pneumatic drive.
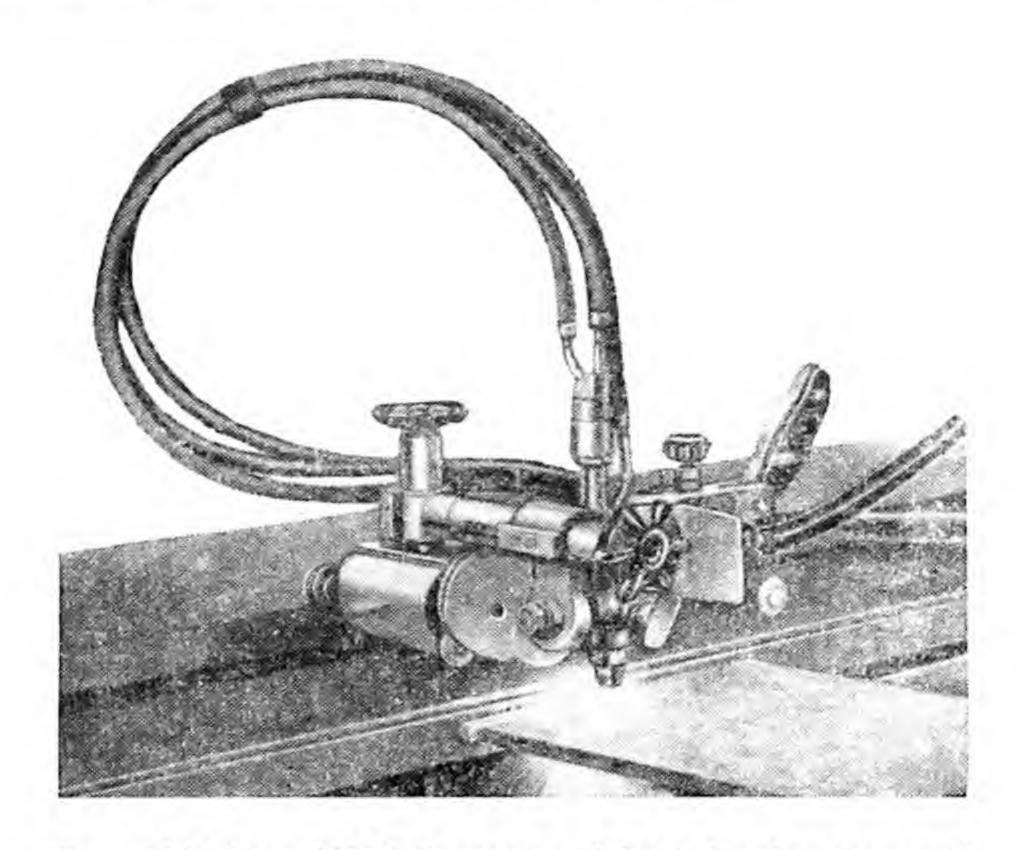

Fig. 145. Type ППК-2 oxygen-driven tractor-mounted cutting machine

The pneumatic drive propels the machine along the line of cut at a uniform speed which can be adjusted continuously within a broad range. Steady operation of the drive requires an air pressure of the order of 3 or 4 kg/cm².

ППВ machines may likewise come with one or two torches, designated the ППВ-1 and the ППВ-2, respectively (Fig. 144).

Their uses include straight-line cuts in steel plate from 5 to 100 mm thick, cilcular cuts up to 540 mm and more in diameter, and the cutting of gentle profiles where electric-driven machines cannot be employed.

Another machine developed by the Autogenous Welding Research Institute is the ППК-2 (Fig. 145), which is driven by the cutting oxygen jet and is intended for service under field conditions.

(2) Stationary Cutting Machines. Relevant U.S.S.R. standards class stationary cutting machines into (a) straight-line or pantosec type (designated МРЛ); (b) rectangular-co-ordinate type (designated МРК); (c) polar-co-ordinate or camograph type (designated МРШ); and (d) pantographic or oxygraph type (designated МРП).
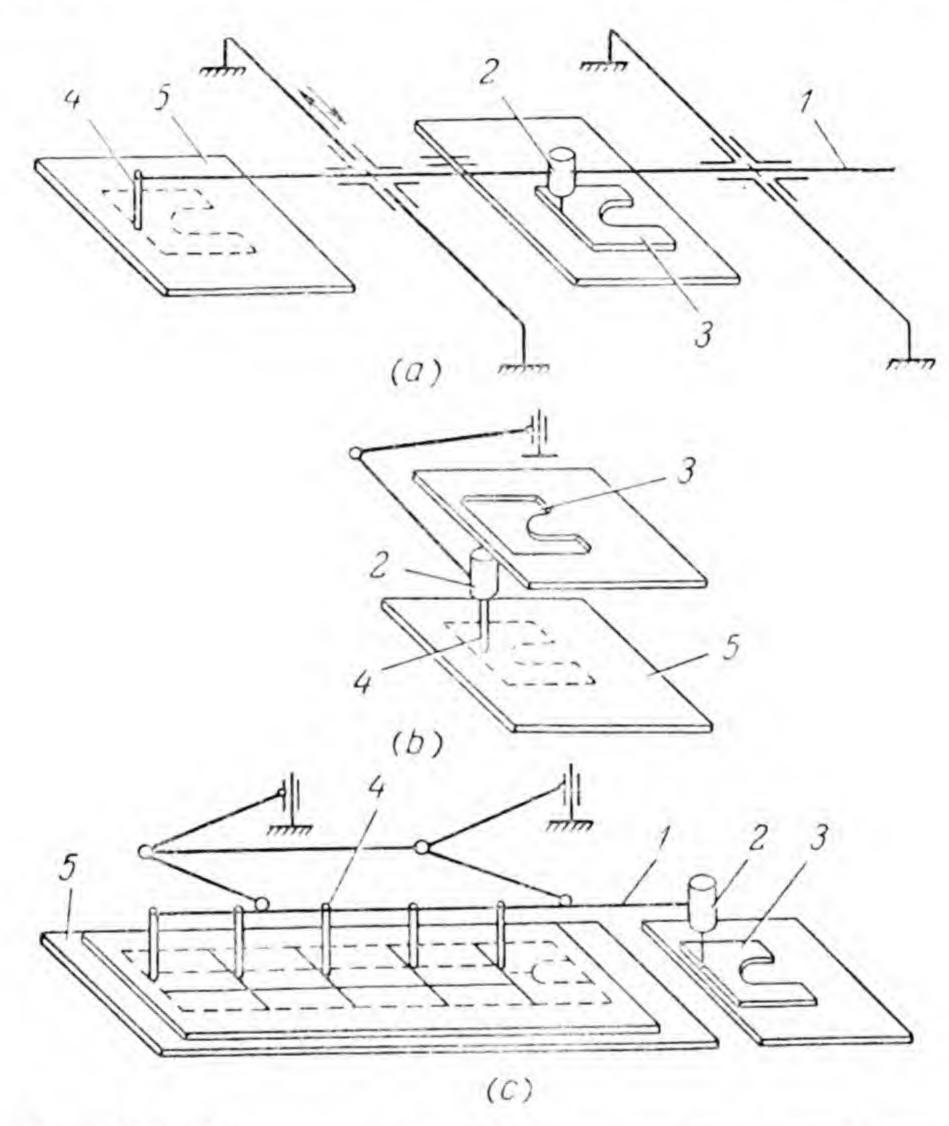

Fig. 146. Kinematics of stationary cutting machines: a — rectangular-co-ordinate cutting machine; b — polar-co-ordinate (camograph) machine; c — pantographic machine; 1 — arm to carry the torch and tracing head; 2 — tracing head; 3 — template; 4 — torch; 5 — cutting table

МРЛ machines serve to cut plate for fabrication purposes and make rectangular parts. The most commonly used within this class are ПР-2.5 and ПР-3.5 machines capable of cutting plate 2.5 and 3.5 m wide, respectively.

As in МРЛ machines, МРК machines cut the desired contours owing to the combination of two motions which are at right

angles to each other. These are the motion of the carriage and the motion of the bar carrying the tracing device and torch. For a schematic diagram of an МРК machine see Fig. 146a.

Machines within this class can make shape cuts to a template, square straight-line cuts, and bevel cuts for subsequent welding. They may use one or several torches; in the latter case, they can cut as many identical shapes or make several straight-line cuts at a time. Machines of МРК classification are most common in Soviet cutting practice and include automatic АСП-1М, СГУ-1-58, МДМ-2 and МДФКС machines.

In МРШ (polar-co-ordinate or camograph-type) machines, the torch is guided around the contour by two frames hinged together. One of the frames is hinged on a stationary column; the
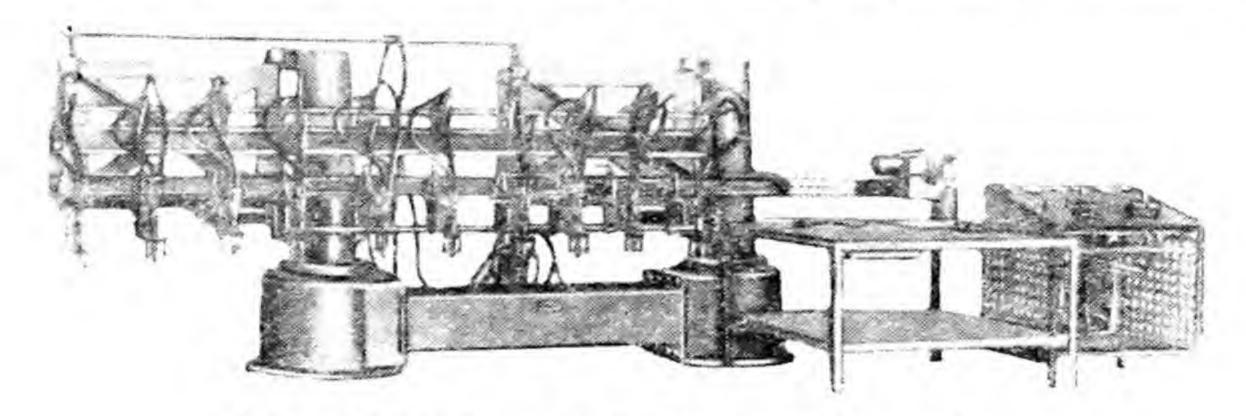

Fig. 147. Type M-10-1 pantographic machine

other carries the tracing wheel and torch. As a rule, the torch is placed beneath and in line with the tracing wheel. For a schematic diagram of a polar-co-ordinate cutting machine see Fig. 146b. Included in this class are the stationary АСШ-2 machine and the portable АПШ-1 machine.

A machine in the МРП (pantographic or oxygraph) class has two jointed systems each incorporating two jointed beams. The jointed systems are connected by two parallel bars of equal length, thereby making up two jointed parallelogram frames. The tracing device is mounted on the outer bar which connects the ends of the outer frames; the outer bar may also carry several torches for cutting several identical parts at a time. A schematic diagram of a pantographic machine is shown in Fig. 146c. In the Soviet Union, pantographic machines have not found any appreciable use. The only machine in this class is the ten-torch M-10-1 unit (Fig. 147).

The tracing mechanisms employed in stationary cutting machines may be of the mechanical, electromagnetic, or photocell type and may be placed locally or remotely. In the latter case,

use is usually made of electromagnetic and photocell tracing heads.

Mechanical tracing heads are usually moved by hand around a drawing or the contour laid out on the plate, and the torch follows the desired profile. This type of tracing device is employed in the АСП-1 co-ordinate cutting machine (Fig. 148). The head has a knurled guide roller *1* which travels over the tracing table and moves the torch-carrying arm in the desired direction. When cutting from drawing, the roller is guided by a handwheel *2*.
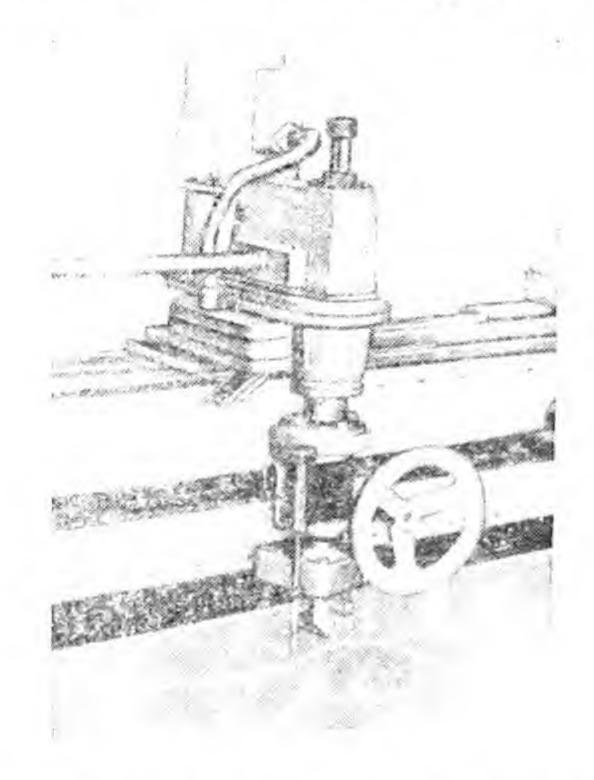

Fig. 148. Mechanical head to Type АСП-1 rectangular-co-ordinate cutting machine

For higher accuracy of tracer motion, the head has an index *3* whose point is directed along the drawn profile. In making longitudinal straight-line cuts, the guide roller is set by the handwheel to move along the main track, and the cross-carriage is locked.

In making transverse straight-line cuts, the cross-carriage is made free to move, the main carriage is locked, and the guide roller is adjusted to travel at right angles to the main track.

In cutting to a profile laid out directly on the work, the torch-carrying arm or carriage and the main carriage are left unlocked, and the torch is guided by means of the handwheel.

In addition, the mechanical tracing head may be employed to cut circles. In such a case, the centre point of a radial bar is mounted on the tracing table of the machine, and the tracing head attached to the radial bar follows the circle of the set radius.

Magnetic tracing heads depend for their operation on steel templates. The key part of a magnetic tracing head is a steel

knurled magnetised roller driven by an electic motor. Being magnetised, the roller adheres to the steel template as it advances automatically around the contour of the steel template. The torch rigidly coupled to the tracing head describes exactly the same motion as the magnetised roller, thereby cutting the shape
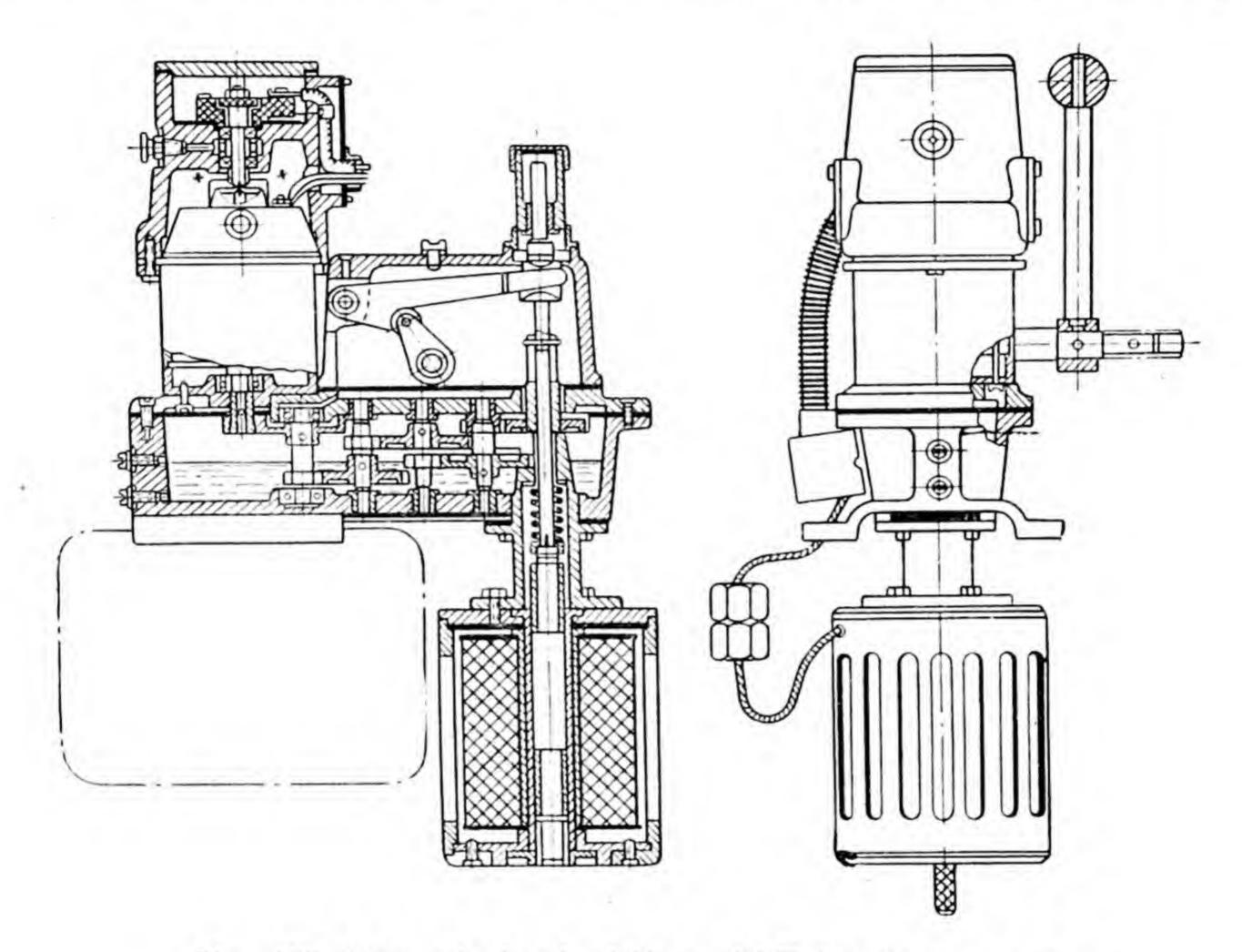

Fig. 149. Magnetic head of Type ACΠ-1 cutting machine

traced. The magnetic tracing head employed in the ACΠ-1 machine is shown in Fig. 149.

A serious drawback of magnetic tracing heads is that they require a large number of steel templates which adds to cutting costs and take up storage space.

These drawbacks are nonexistent in the case of photo-electric tracing heads.

Automatic photo-electric tracing from full-size drawings may be employed on any flame-cutting machine. The best results, however, are obtained on rectangular-co-ordinate machines fitted with a mechanical tracing assembly and a photocell scanning unit. The scanning unit automatically follows the contour of a

drawing and guides the mechanical head which is rigidly coupled to the torch.

The photo-electric tracing systems employed commercially in the Soviet Union are classed into the amplitude type and the impulse type.

In amplitude-type systems, use is made of drawings in broad black lines. The light spot is made to follow one of the sides of the drawing line. No control voltage is generated when the centre of the light spot is accurately placed on the side of the line (Fig. 150).
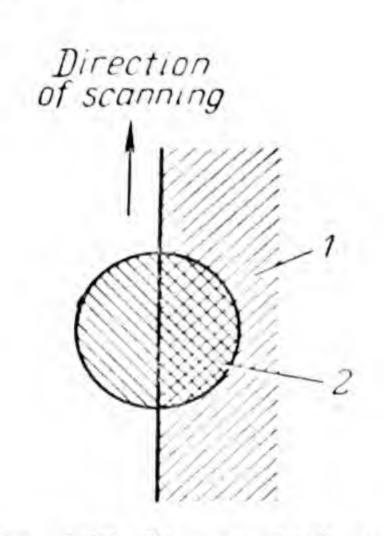

Fig. 150. The neutral position of the light spot in the amplitude-type scanning system:
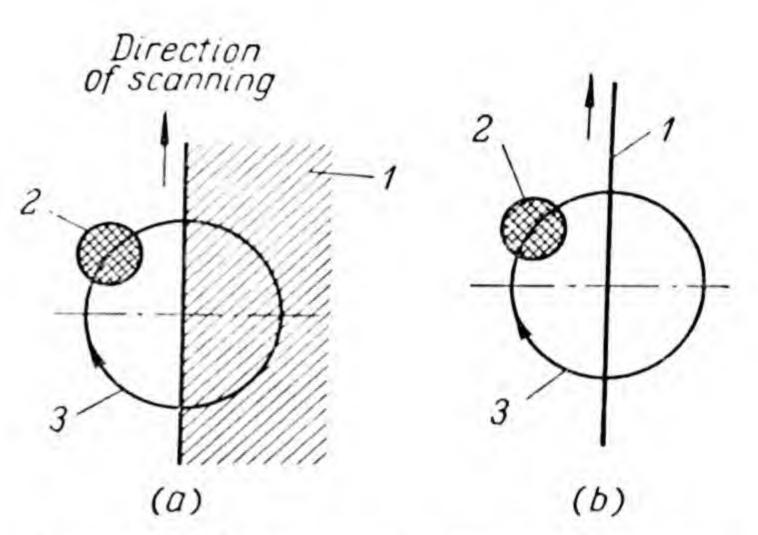1 — drawing line; 2 — light spot

Fig. 151. Motion of the light spot in the impulse-type line-follower system in the neutral position:
a — broad drawing line; b — fine drawing line; 1 — drawing line; 2 — light spot; 3 — circular trace

In impulse-type systems, contour drawings may use both heavy and fine lines. In addition to the motion along the line, the light spot is made to move in a circle (Fig. 151a and b). As the light spot describes the circle with a speed which is many times as high as the rate of advance, the trace of the light spot on the drawing is assumed to be the shape of a regular circle. No control voltage is generated when the centre of the circular trace is on the side of a broad contour line or in the centre of a fine contour line.

Fig. 152 illustrates the principle of an impulse-type line-follower system. The scanning unit has a lamp 1 which sends out a light beam which is reflected from a stationary inclined mirror 2 into an eccentric lens 3 driven by an electric motor 4. The lens focuses the light beam into a spot on the drawing. The photocell 5 is enclosed in a box and can only pick up the light rays reflected from the white background of the drawing. When the light

spot is on the black line, the photocell picks up practically no light. The light spot crosses the drawing line at a rate equal to the frequency of the a-c mains, and the photocell generates pulses of photocurrent. Amplified by a valve amplifier *6*, these pulses are fed to the grids of two thyratrons *7* so connected that one of them is conducting during the positive half-cycles and the other during the negative half-cycles.

The difference current of the two thyratrons drives the steering motor *8* which rotates the scanning unit *9* so that the light
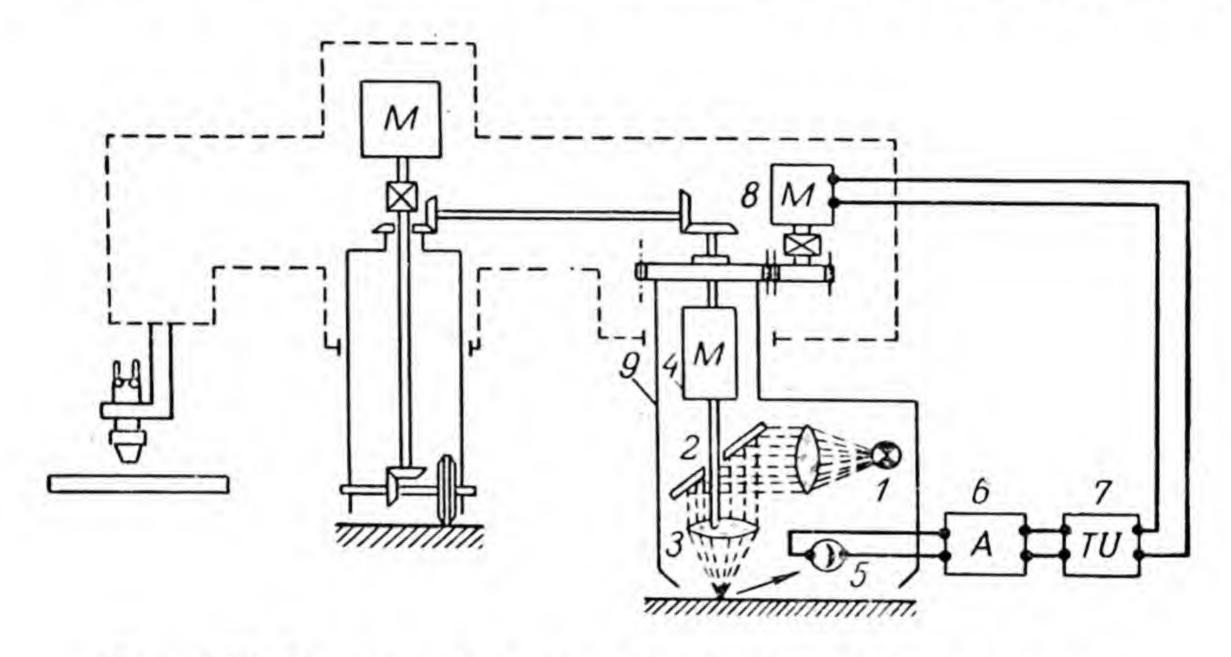

Fig. 152. Diagram of a photo-electric line-follower system

spot is made to follow the drawing line accurately. The guide roller of the mechanical head describes exactly the same motion.

Of course, this is only a simplified description of how a photo-electric line-follower system operates. Actual systems are more sophisticated. One of them, the amplitude-type photocell attachment to МДМ-2 automatic cutting machines, incorporates a photocell scanning unit, a system of lenses, a mechanical tracing assembly, a steering reduction gear and a control cabinet. This system operates as follows.

The tracing assembly transmits the drive from the cutting machine motor to the forward drive wheel which runs over the drawing.

The optical system throws a circular light spot onto the drawing. Reflected from the drawing, the beam of light is picked up by a photo-sensitive resistor connected as an arm of a bridge in the scanning unit. The scanning unit is so adjusted that when the light flux reflected from the white background of the drawing

is equal to that reflected from half the area of the light spot, the bridge is at balance. As the light spot moves away from the drawing line, the resistance of the photo-sensitive resistor changes, and the balance of the bridge is upset. As a result, what is called error or control voltage is derived from across the bridge to energise the servomotor which drives the steering reduction gear. The latter rotates the light spot about the axis of the photocell with a preset eccentricity until the light spot is again put on the drawing line and the bridge is again at balance. At the
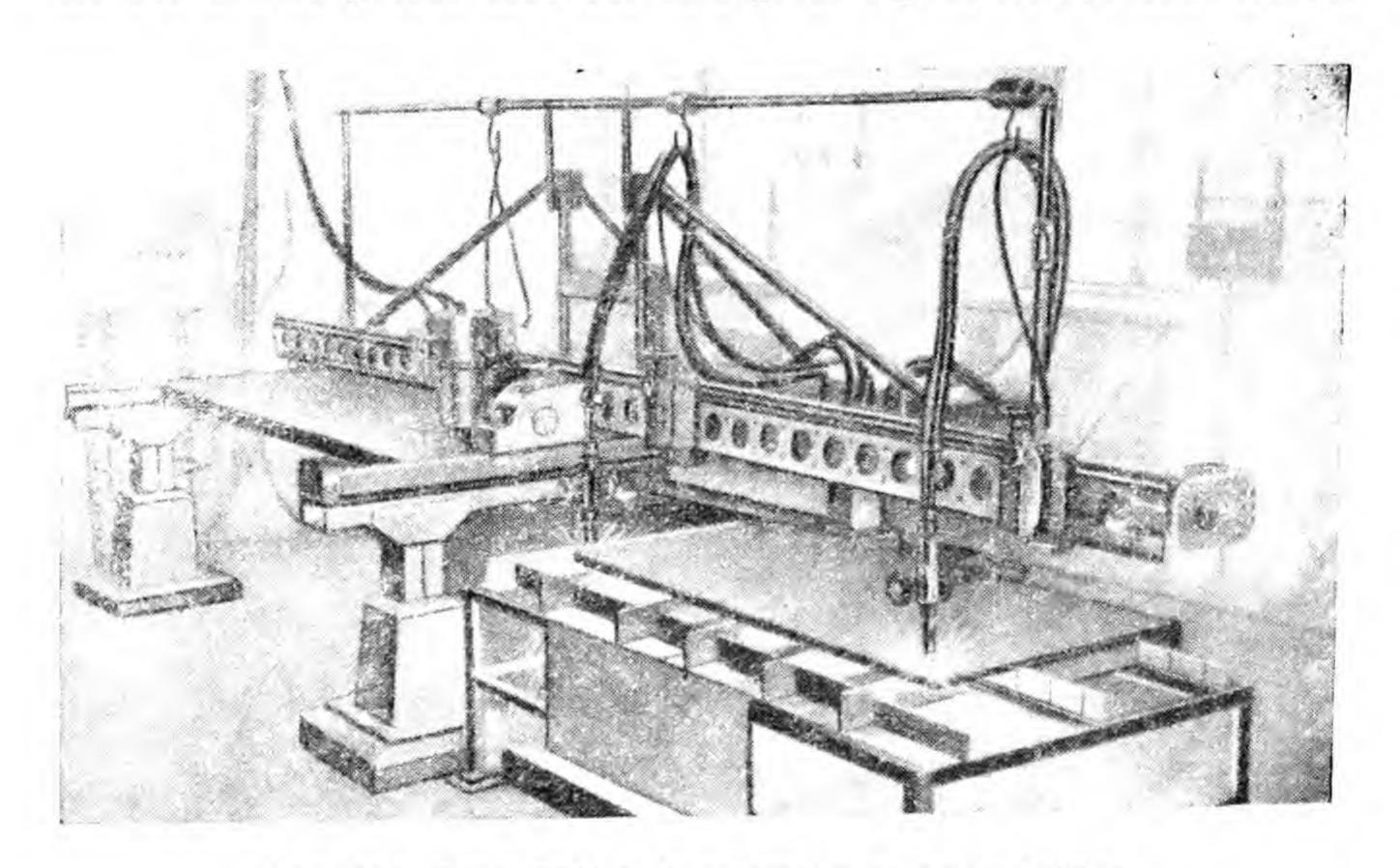
Fig. 153. Type СГУ-1-58 automatic cutting machine

same time the steering motor guides either the longitudinal or the cross-carriage, as the case may be, at a preset speed, and the same profile is described by the torch on the cutting table of the machine.

This photocell attachment is also employed in the СГУ-1-58 automatic machine developed by the Autogenous Welding Research Institute in 1958 (Fig. 153) and in the МДФКС rectangular-co-ordinate machine. The former is a rectangular-co-ordinate machine with a movable tracing table and a stationary cantilever arm on which up to four torch carriages may move. The latter incorporates a variable-scale line follower — one of the most sophisticated tracing systems using both magnetic head for

work from templates and a photocell scanning unit for work from drawings.

In the МДМ-2 machine which has six torch carriages, this system makes possible to cut six identical parts to variable scales of 10 : 1, 5 : 1 and 2 : 1.

The machine consists of a control (or tracing) unit and a controlled (or cutting) unit (Fig. 154) connected only electrically. The control (tracing) unit may be built into a separate glazed booth.
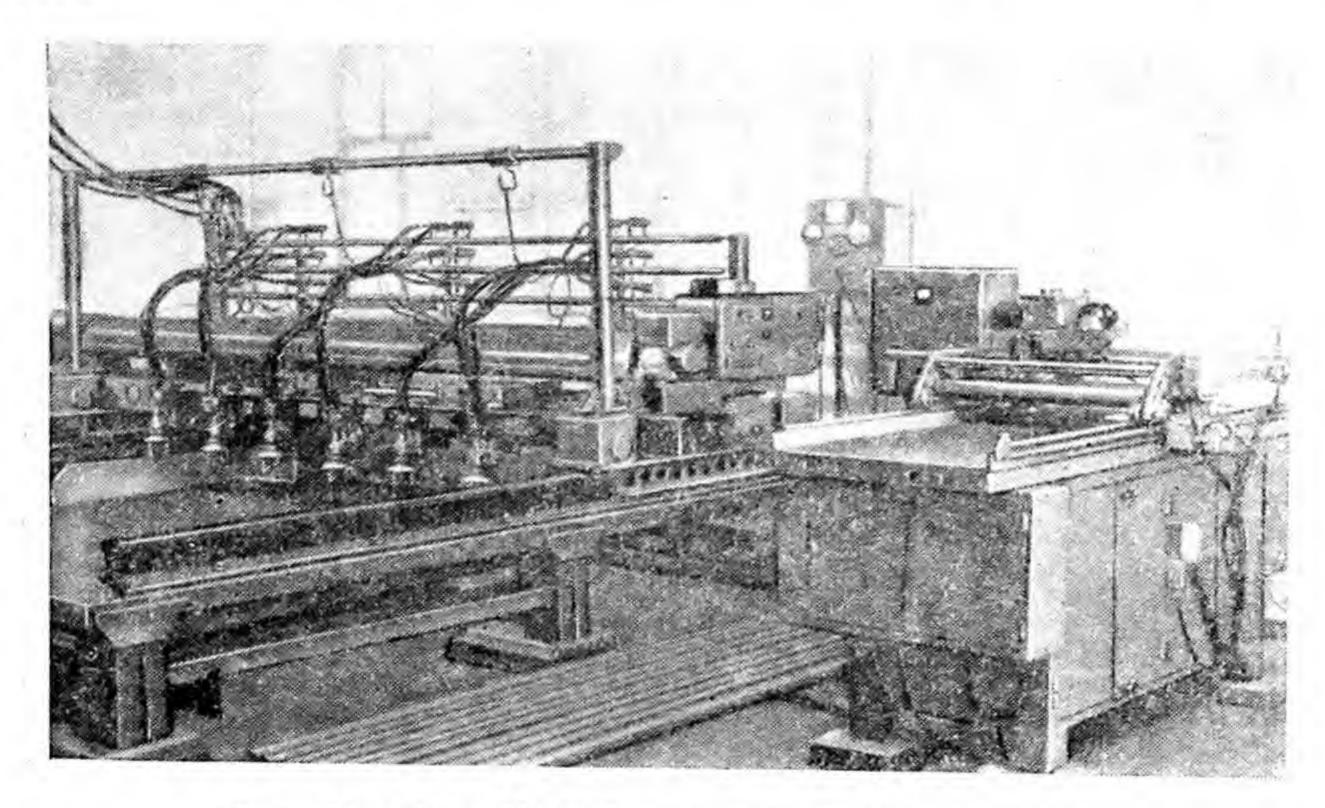
Fig. 154. Type МДМ-2 automatic cutting machine

The tracing unit, which is of the rectangular-co-ordinate type, consists of a substantial carriage *1* mounted on wheels *6* and free to travel on tracks *2* (Fig. 155). The main carriage *1* supports another carriage fitted with two bars *3* to which a driving mechanism *4* is made fast. The bars are free to move in slides *5*. The main carriage *1* and the bars *3* move at right angles to each other.

As the magnetised roller of the driving mechanism *4* moves around the template, the main carriage and the bars also move, describing the contour of the template. The tracing unit resolves any motion of the magnetic tracer in a plane into linear motions along the axes of rectangular co-ordinates.

One of the wheels *6* of the main carriage is permanently coupled to the rotor of a selsyn *S1*; and the latter will move through an angle in proportion to the movement of the main carriage.

Similarly, the travel of the bars *3* will rotate the rotor of another selsyn *S3*, as the latter is coupled to one of the bars by means of a wheel *7*. The stators of the two selsyns are rigidly mounted on the main carriage *1*.

The controlled (or cutting) unit of the machine, driven by two servomotors *M*, is in fact a mirror image of the tracing unit.
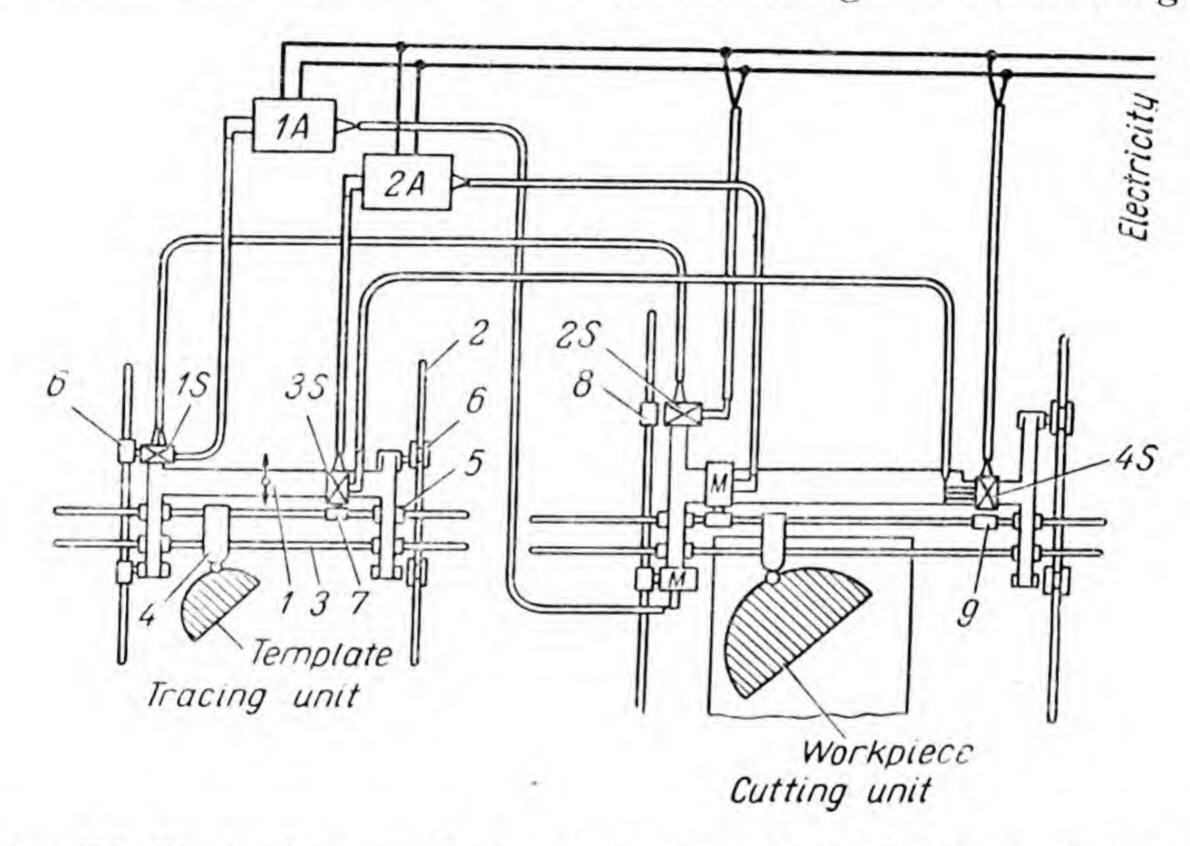
Fig. 155. Block diagram of the variable-scale line-follower employed in МДМ-2 cutting machines

The selsyns of the tracing and cutting units are connected in pairs (*S1* and *S2*; *S3* and *S4*), each pair being connected into what is known as the control-transformer circuit. In this case, the three-phase (stator) windings of each pair are connected with each other, while the signle-phase (rotor) winding of the selsyn in the cutting unit is fed with alternating voltage, and the single-phase (rotor) winding of the selsyn in the tracing unit generates an alternating voltage whose magnitude depends on the relative position of the two rotors, and whose phase depends on the direction in which the two rotors are turned away from each other.

The error voltages taken from across the rotor windings of the selsyns *S1* and *S3* are fed to amplifiers *A1* and *A2*, each consisting, in a general case, of:

(a) a voltage amplifier to amplify the amplitude of the error voltage;

(b) a discriminator, to determine the phase of the error voltage;

(c) a power amplifier, to amplify the power of the error signal.

The amplified error voltage is impressed on the control or servomotors *M* in the controlled (cutting) unit. When the rotors of the respective selsyns take up a similar position, there is no control voltage.

Referring to the block diagram in Fig. 155, electrical connection of the tracing and cutting units is by two independent, though identical channels, each representing one of the two coordinate axes.

In a general case, the selsyn tracing system operates as follows. As the driving mechanism travels around the desired profile, the rotors of the selsyns *S1* and *S3* are displaced through an angle, while the rotors of the selsyns *S2* and *S4* remain stationary for some time initially. This brings about an error voltage corresponding both in magnitude and phase to the difference in the relative position of the rotors in the paired selsyns. After it is amplified by the amplifiers *A1* and *A2*, the error or control voltage energises the servomotors *M* in the cutting unit. The servomotors rotate to actuate the respective parts of the cutting unit, and the rotors of the selsyns *S2* and *S4* are displaced through respective angles until the error voltage is reduced to zero.

The control voltage impressed on the servomotors *M* is reduced as the difference in the relative position of the rotors in the paired selsyns is minimised. As a result, their speed is decreased, and so is that of the torches in the cutting unit. This, again, increases the difference in the relative position of the rotors in the paired selsyns, and the cycle is repeated all over again. Thus, the selsyn system continuously controls the speed of torch travel as a function of the speed of the tracing head.

As is the case with any continuous (functional) control process, the magnitude of the controlled variable (the position of the torch) is made to oscillate around the magnitude of the manipulated variable (the position of the magnetic tracer or the light spot with respect to the contour).

As long as the driving mechanism continues to move, the rotors take up positions which differ in proportion to the rate of change in the speed and direction of the mechanism's travel along the co-ordinate axes of the tracing unit and to the structural properties of the entire machine. Obviously, there will be no difference in the relative position of the rotors when the driving mechanism stops.

The rotors of the paired selsyns in the tracing and the cutting units rotate both in synchronism and in phase. For this reason, the movements of the units along the co-ordinate axes will be proportional both in the distance covered and in the speed maintained. Consequently, the outline described by the torch will be an exact scaled-down or scaled-up replica of the contour described by the tracing head (provided, the same proportionality is maintained for both co-ordinate axes).

Referring to Fig. 155 and summing up, it may be written that

$$M_x = \frac{D_9}{D_7} \; ; \; M_y = \frac{D_8}{D_6} \; ;$$

where M_x and M_y are the tracing scale factors for the X and the Y axes respectively;

D is the diameter of wheels 6, 7, 8 and 9 in the tracing and cutting units of the machine.

If $M_x = M_y = M$, this quantity will determine the similarity of the contours described by the tracing head and the torch, i. e., the scale of tracing.

The scale of tracing may be changed at will by varying the gear ratio between the respective wheels and the selsyn rotors, which is done with the aid of change gears.

The above system makes it possible to scale profiles up or down, using both drawings and templates. Another advantage is that amplification permits the use of low-power servomotors. The МДМ-2 machine, intended primarily for cutting large parts, uses small-size templates when the tracing is done by the magnetic head and scaled-down drawings when the photocell does the job.

(3) Templates for Magnetic Tracing. In machine cutting from templates, the precision of shape and dimensions is mainly dependent on the quality of the template used. Templates may be designed for use with guide rollers of either the tension type or of the magnetic type, both types differing in the way they engage the template.

Templates for use in conjunction with tension guide rollers were employed in early makes of cutting machines, like the AC. They were made from brass or aluminium strip 30 mm wide and 2.5 mm thick. Modern cutting machines do not use tension-type guide rollers and strip templates.

Templates for use with magnetic guide rollers are usually fabricated from low-carbon steel (such as Cт. 2 or Cт. 3) 5 to 10 mm thick. In some cases, they are made from wood, with the profile face lined with steel strip.
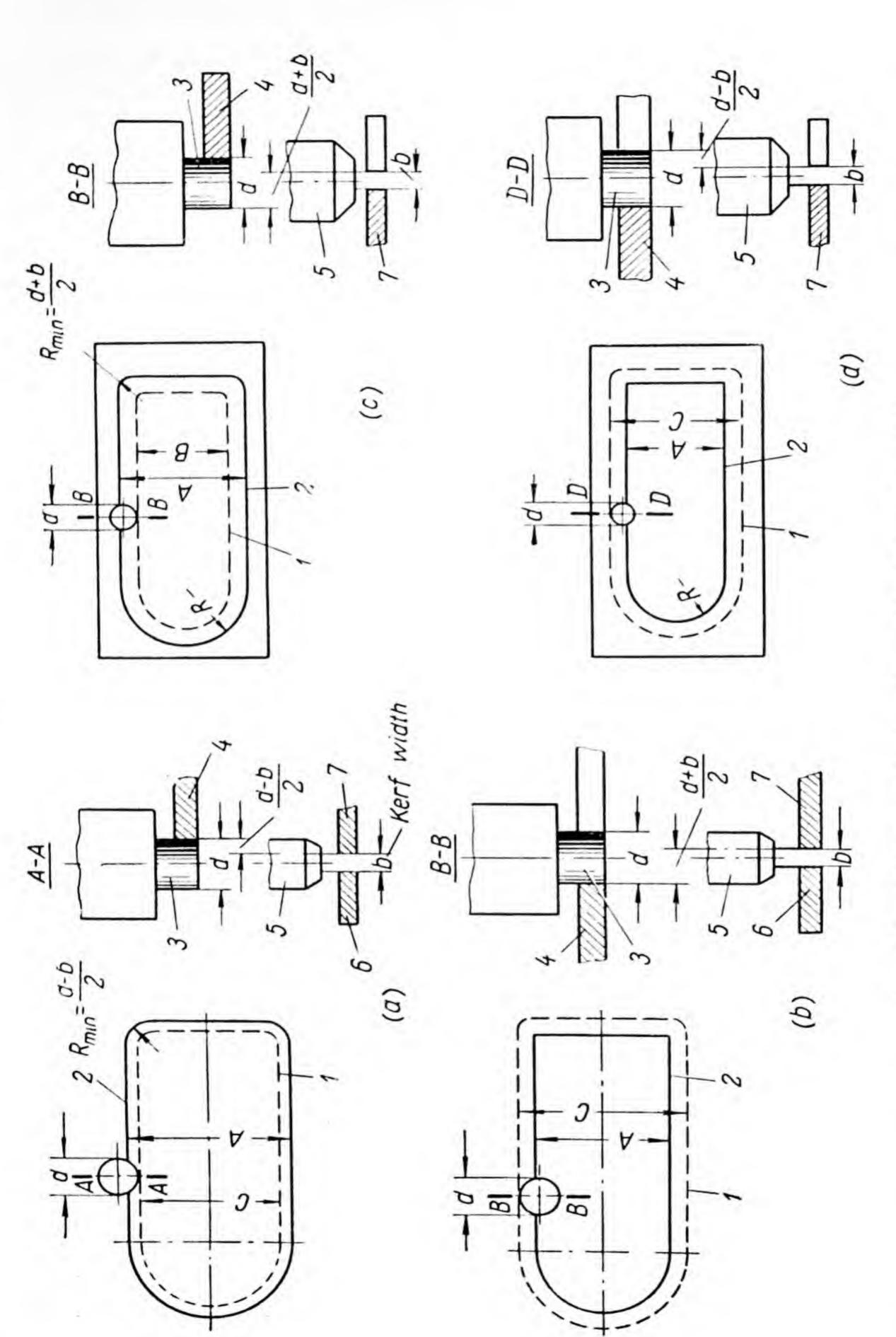
Fig. 156. Relative size of a template and a workpiece:

a — the part is cut and the template is traced around their outside peripheries; *b* — the part is cut around its outside periphery while the template is traced around its inside periphery; *c* — the part is cut around its inside periphery while the template is traced around its outside periphery; *d* — both the part is cut and the template is traced around their respective inside peripheries: *1* — template contour; *2* — part contour; *3* — magnetic follower; *4* — template; *5* — tip; *6* — scrap; *7* — part to be cut

Four techniques of profile tracing are possible, namely:

(a) both the part is cut and the template is traced around their respective outside periphery;

(b) the part is cut around its outside periphery while the template is traced around its inside periphery;

(c) the part is cut around its inside periphery while the template is traced around its outside periphery;

(d) both the part is cut and the template is traced around their respective inside periphery.

Case "a": The dimensions of the template (Fig. 156a) are determined by the equation:

$$B = A - (d - b), \qquad\qquad\qquad \text{(XXII.1)}$$

where A = size of the part;
B = part of the template matching the part of size A;
d = diameter of the magnetic guide roller;
b = width of the cut.

The minimum radius of curvature for the cut edges is

$$R_{min} = \frac{d - b}{2}.$$

Case "b": The dimensions of the template (Fig. 156b) can be determined by the equation:

$$B = A + (d + b). \qquad\qquad\qquad \text{(XXII.2)}$$

The minimum radius of curvature for the cut edges is zero ($R_{min} = 0$).

Case "c": The dimensions of the template (Fig. 156c) can be found by the equation:

$$B = A - (d + b). \qquad\qquad\qquad \text{(XXII.3)}$$

The minimum radius of curvature for the cut edges is

$$R_{min} = \frac{d + b}{2}.$$

Case "d": The dimensions of the template (Fig. 156d) can be determined by the equation:

$$B = A + (d - b). \qquad\qquad\qquad \text{(XXII.4)}$$

The minimum radius of curvature for the cut edges is zero ($R_{min} = 0$).

4. Machine Cutting Practice

In the shape cutting from templates the most crucial stage is the start of a cut, for it is in this stage that most of the defects, such as rugged spots, fused edges and spatter occur. In cutting

with one torch these defects may be avoided by (a) using an ancillary template (mainly when templates are held in place by magnetic clamps) and (b) by starting the cut in scrap and by moving the torch by hand until the magnetic guide roller comes in contact with the template.

In effect, the purpose of an ancillary template is likewise to enable a cut to be started away from the edge. It is therefore
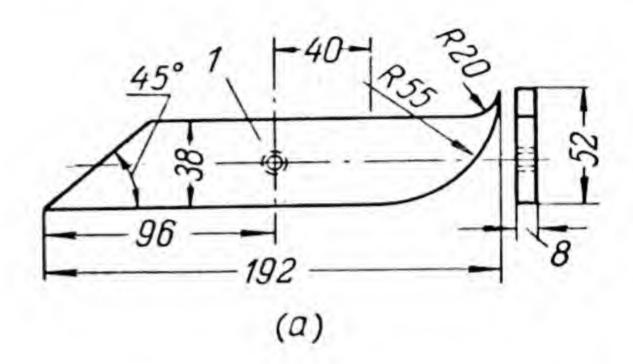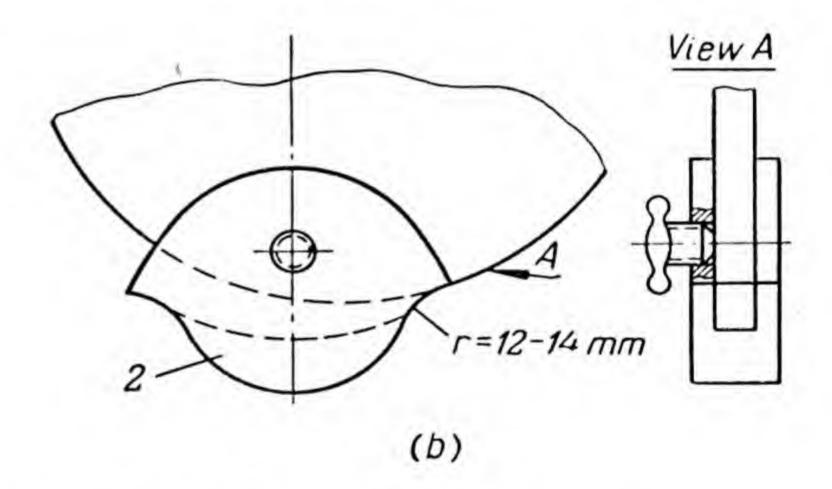
Fig. 157. Ancillary templates

placed over the main template so that there is no sharp changes in outline between the two.

Before starting the machine, the magnetic roller is brought in contact with the ancillary template. When the machine is started, the torch makes a "run-in" cut through the scrap until the magnetised roller goes smoothly over onto the main template, the ancillary template is removed, and the torch does further cutting around the main template. Fig. 157a and b shows some of the types of ancillary templates.

In case "b" the cut is started and the magnetised roller is positioned away from the edge of the template (within about 20 to 30 mm). After the "run-in" cut is started, the motor is turned on to drive the guide roller, and the magnetic head and torch together are moved by hand towards the template until the roller comes in contact with the latter. As soon as the roller touches

the template, the magnetic coil of the driving mechanism is energised.

Both starting techniques produce a smooth finished cut unmarred at the starting point.

When cutting with a single torch from a drawing, the starting procedure is as follows. In the case of machines fitted with a mechanical head having an index, the torch with the oxygen turned on is positioned above the start of a cut (usually above the edge of the piece), and the guide roller is lowered onto the tracing table so that the index is outside the contour (within 20 or 30 mm). After the cut is started, the motor is switched on, and the guide roller is directed by means of the handwheel on the mechanical head towards and then around the drawn profile, thereby guiding the torch as required.

When cutting with a single torch to the profile laid out on the piece, in machines having a mechanical head, the torch, with the preheating flame turned on, is positioned above the start of the cut, and the guide roller is brought down onto the tracing table. After the cut is started, the torch is directed by means of the handwheel located near the torch (or on the torch arm as in the АСП-1М machine) towards and then around the laid-out profile.

The starting technique for circular cutting with the aid of a radius rod in machines equipped with a mechanical head is as follows. The radius bar is inserted into the recess in the mechanical head, the adjustable centre point is placed on the tracing table, and the desired radius is set by shifting the centre point along the rod. The piece and the centre point are checked for relative position, the latter is affixed to the tracing table by a method varying from machine to machine (say, by means of magnetic clamps), and the guide roller is lowered onto the tracing table. Rotation of the handwheel brings the torch at the start of the lead-in cut, the cut is started, the torch is led onto the circle, the centre point of the radius centre is clamped, the motor is switched on, and further cutting is done as guided by the radius rod.

Ancillary templates can also be used to obtain smooth finished cuts unmarred at the starting point in quality cutting by several torches, as the "lead-in" cut is likewise made away from the contour. A very important point in multitorch quality cutting is that the torches should be positioned at right angles to the piece surface. In multitorch machine strip cutting, the torches are positioned before the cut is started over the edge of the piece, and the "floating" rollers which maintain constant the distance between the tips and the metal surface are placed on the work-

piece At the end of the cut, just as the rollers are about to come off the piece, they are braked, and the cut is completed without them.

5. Bevel Cutting

Edge preparation for welding by flame cutting includes single bevels with a root face made by two torches and double bevels made by three torches. This job can be performed with equal efficiency by rectangular-co-ordinate and pantographic stationary machines and by portable two-torch and three-torch motor-driven machines. Machine-cut bevels have high surface finish and require no subsequent machining, the only requirement being des-lagging.

(1) Single Bevels. In making single bevels, one torch is positioned at a right angle to the surface of the piece, and the other at the desired angle of bevel. Two techniques may be employed for cutting single bevels.

1st technique: The torches are set distance *A* apart in the direction of cutting (Fig. 158*a*). The vertical torch leads the inclined torch and produces a square cut which is essential for the proper root face to be obtained. The inclined torch is set back from the line of cut by a distance *B* and produces the desired bevel. The separation between the torches (distance *A*) varies with metal thickness and should be such as to prevent the slag from sticking to the bottom of the cut. Distance *B* depends on metal thickness, bevel angle and root face.

Table 46

Single-bevel Cutting for Welding

Metal thickness, mm	Tip size	Cutting jet separation, mm	Pressure, atm (gauge)		Average cutting speed, mm/min
			oxygen	acetylene	
10	1	30	4.5	0.02-0.5	485
20	2	25	4.5	0.02-0.5	395
30	3	20	4.5	0.02-0.5	330
40	3	15	5.5	0.02-0.5	290
60	4	10	6	0.02-0.5	230
80	4	8	7.5	0.02-0.5	190
100	4	6	10.5	0.02-0.5	165

Note: The table applies to steel containing not over 0.3 per cent carbon, cut with oxygen of at least 90 per cent purity and using oxygen hoses 9-10 metres long and 9.5 mm inside diameter.

2nd technique: As in the former case, the vertical torch leads the inclined torch by distance *A* (Fig. 158*b*). Distance *B* is much shorter, and as much metal has to be cut off as in the previous case. Table 46 suggests the cutting conditions for the second technique.
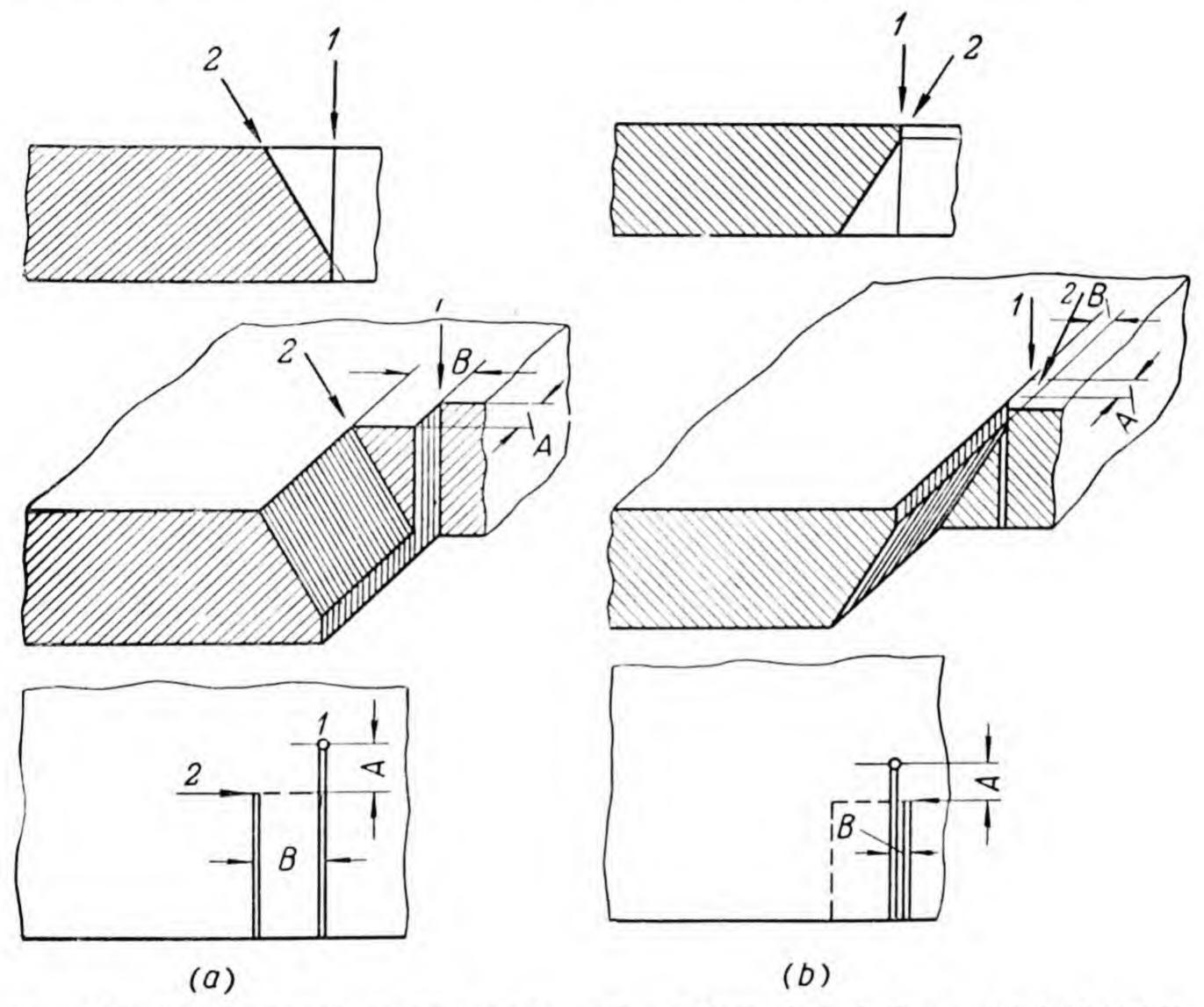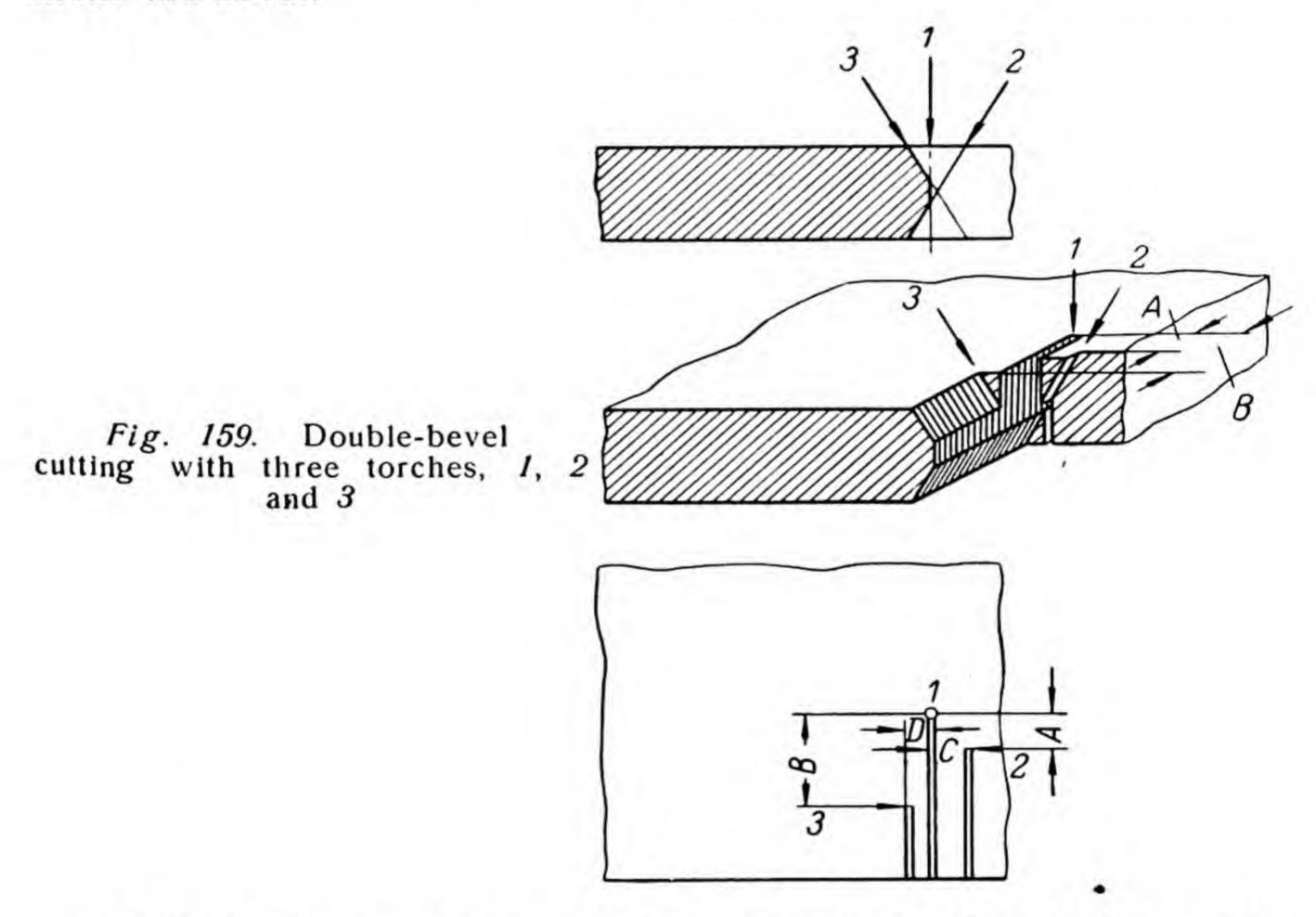
Fig. 158. Single-bevel cutting with two torches, *1* and *2*: *a* — 1st technique; *b* — 2 nd technique

The advantages of the second technique are that the bevel cut can be started without stopping the machine and that less slag is formed and it comes off the cut edges easily. Its disadvantage is that less smooth cuts are obtained in steel over 35 to 40 mm thick.

(2) Double Bevels. Double bevels are made with three torches operating simultaneously (Fig. 159). The vertical torch leads by distance *A* the torch making the bottom bevel and by distance *B* the torch making the top bevel. The distance *A* should be long

just enough to separate the oxygen jets. The time separation between the cutting jets of the first and second torches should likewise be small, in order that the metal cannot cool and the vertical cut face cannot be covered by a film of solidifying oxides. If these requirements are not met, the oxygen jet of the third torch will lose some of its kinetic energy in its encounter with the surface oxide and, swerving down the vertical face, will fail to make the bevel.

Fig. 159. Double-bevel cutting with three torches, *1, 2* and *3*

In light gauges, the distance *A* should be 10 to 12 mm. The third torch operates under more favourable conditions as it cuts the metal well heated by the first two torches. For this reason, distance *B* may be much greater than distance *A*. The distances *C* and *D* (Fig. 159) vary with metal thickness, bevel angle and root face.

The arrangement of torches shown in Fig. 159 provides for high production rates and eliminates the necessity of stopping the machine when starting the second and third torches.

Distances *A* and *B* decrease as metal thickness increases and cutting speed is slowed down. If the start of the cut made by the second torch is outside the area heated by the first torch, they both should be positioned in line (in a plane normal to the direc-

tion of cutting), so that they heat one and the same metal area. In addition, the second torch, which is tilted in a plane normal to the cut, is also given a tilt in the direction of cutting so that its jet passes behind that of the vertical torch, including its drag (Fig. 160). This improves the operating conditions for the third
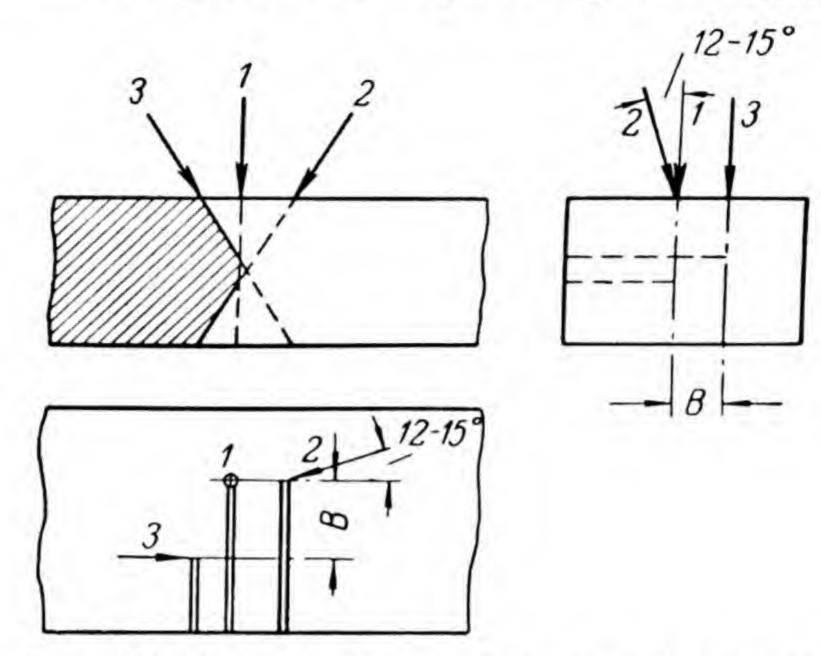
Fig. 160. The position of the second torch in bevel cutting

torch, as the metal is well heated by the first two, and the third torch usually starts its cut without the machine being stopped. The bevel made by the third torch has an especially high surface finish.

The cutting conditions for bevelling with three torches are summarised in Table 47.

Table 47

Double-bevel Cutting for Welding

Metal thickness, mm	Tip size	Distance A, mm	Distance B, mm	Cutting oxygen pressure, atm (gauge)	Average cutting speed, mm/min
20	2	10-12	25	4.5	370
30	3	12-3	22	4.5	310
40	3	2-0	20	5.5	270
60	4	0	18	6	215
80	4	0	16	7.5	190
100	4	0	16	10.5	155

Notes: 1. See Table 46.
2. The pressure of preheating oxygen is 3 kg/cm² in all cases, and that of acetylene 0.02—0.5 kg/cm².

6. High Cutting Rates

There always is one way or another for reducing waste in machine time and auxiliary operations and thereby raising production rate through an efficient layout of the scene of cutting operations.

This is especially true of modern cutting machines equipped with multiflame units and work-positioning facilities. These potentialities may, however, be fully utilised if and when:

(a) a given machine cuts parts within its range of shapes and tolerances;
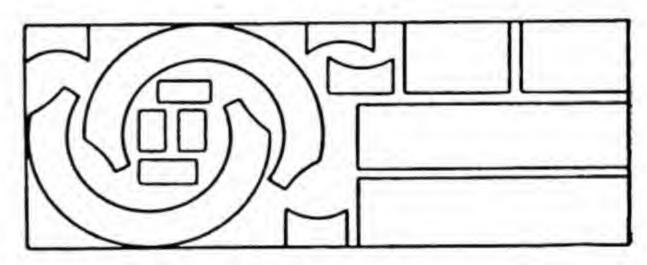
(b) the scene of operation is properly integrated within the entire fabrication process and is equipped with all the necessary materials-handling facilities;

(c) the piece is marked out so as to keep the waste to a minimum;

(d) the sequence of operations is so planned as to reduce idle periods in machine time and to eliminate as many auxiliary operations as practicable.

Fig. 161. Waste-reducing layout for machine cutting with assorted templates

The requirements formulated in (c) and (d) can be met by the use of so-called assorted templates. In using them on, say, a rectangular-co-ordinate machine with magnetic tracing head, several templates are placed on the tracing table in the direction of the transverse motion of the torch. Then, as the cutting operation proceeds, it is possible to cut several parts across the width of the plate without having to move it sideways, with the torch going progressively over from template to template.

After the parts have been cut from all of the templates arranged, the plate or blank is moved the necessary distance lengthways, and another row of parts is cut from the same templates. The best results are derived from the use of assorted templates when the plate is moved alongside the tracing table of the cutting machine on a carriage riding a rail track or on a roller table.

Assorted templates may be of one and the same pattern, i. e., intended for cutting identical parts, or differing both in size and shape.

In the latter case, the waste of metal is particularly negligible as the spaces between the larger parts may be filled in by

the smaller ones. An example of such layout is given in Fig. 161. High production rates are attained with multitorch machines

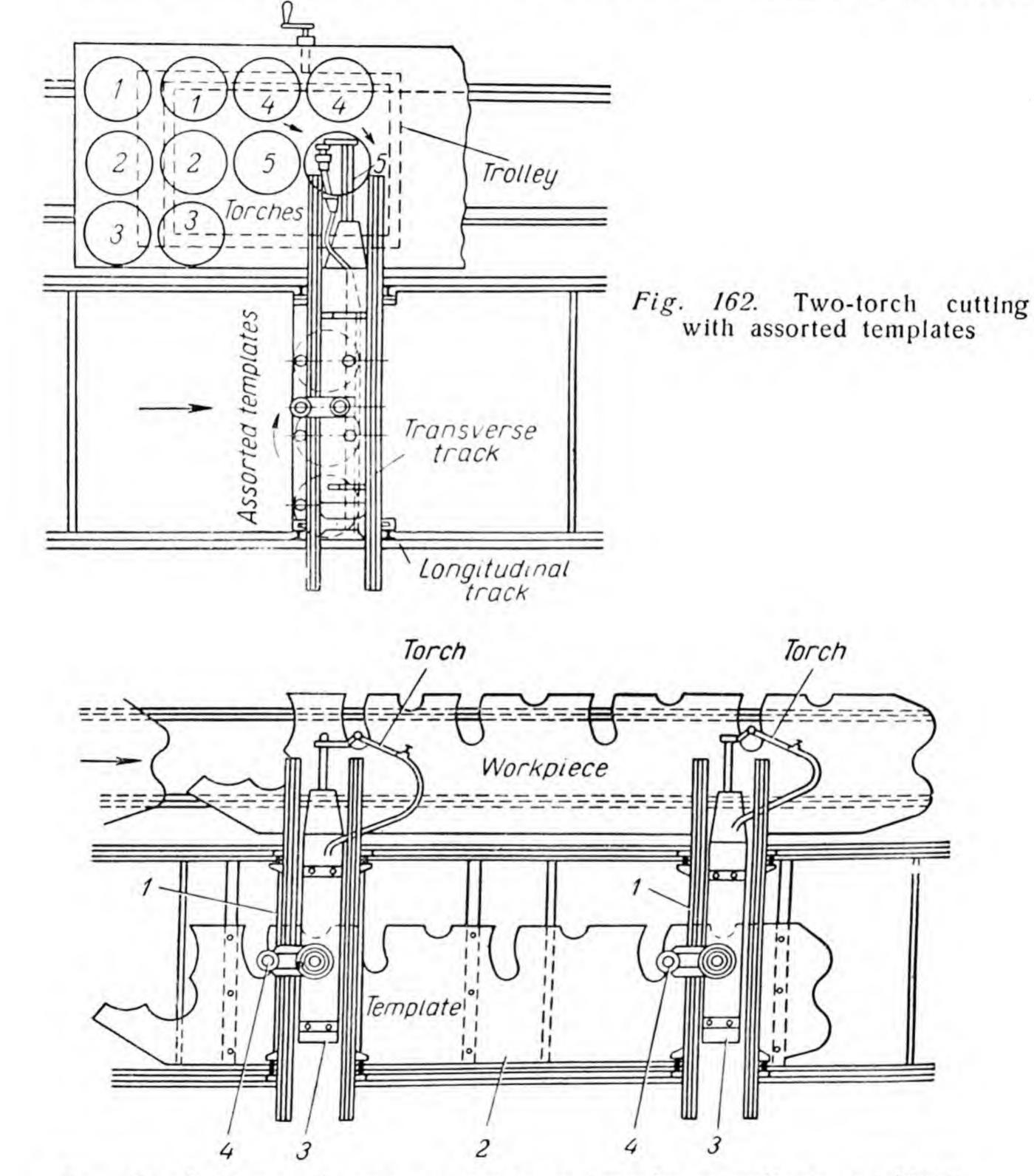Fig. 162. Two-torch cutting with assorted templates

Fig. 163. Cutting with twin automatic rectangular-co-ordinate machines:
1 — main carriage; *2* — template; *3* — cross-carriage; *4* — magnetic tracing head

using assorted templates (Fig. 162) and with twin machines (Fig. 163) in mass-scale fabrication and in the cutting of bulky

pieces. Fairly high production rates are ensured by work-position-ing facilities, in which case identical parts may be cut from a single template.

In addition to the universal types of flame-cutting machines, there exist a large number of special-purpose machines designed to perform a limited number of cutting operations. This group includes the ПMP-600 unit developed by the Autogenous Welding
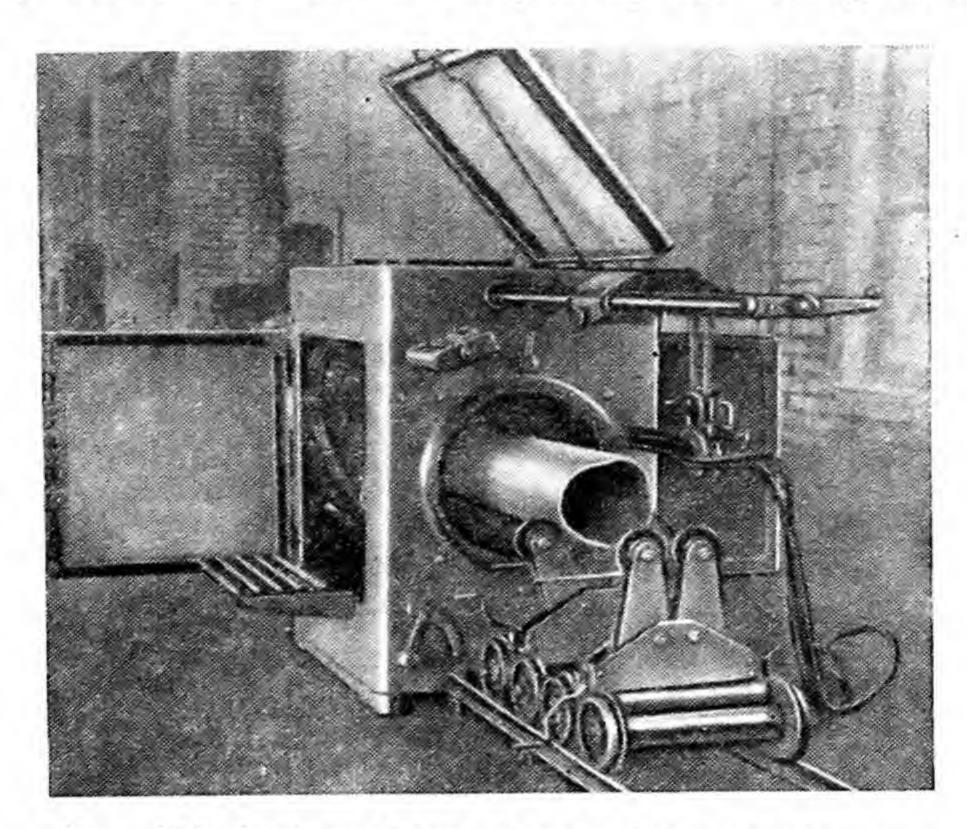
Fig. 164. Link-gear-type pipe-cutting machine de-veloped by Kudryavtsev

Research Institute specifically for heavy cuts in steel and operat-ing on reduced oxygen pressure; a unit for cutting continuously-cast steel ingots to measured lengths; the MPBП-1 machine for cutting circular holes in a vertical plane; a variety of pipe, flange and shape cutting machines; and fixtures for trimming stamped shell plates, etc.

Also belonging to this group are the Kudryavtsev link-gear-type pipe-cutting machine (Fig. 164), and the УBO-1 machine developed by the Bauman Technical School in Moscow for cutting circular holes in medium-pressure boilers.

Where repetitive work is of simple character, production rate can be raised by the use of ordinary jigs and fixtures.

FLAME-CUTTING PRACTICE

1. Cutting Procedure

(1) Surface Cleaning. The surface of the piece should be thoroughly cleaned of mill scale, rust, paint and dirt. Rust and dirt may be removed with a wire brush, while mill scale and paint by what is known as flame blasting or cleaning with a special torch or with a welding or cutting torch, covering a width of not over 30-50 mm along the line of cut.

(2) Starting Flame Cuts. The preheating flame is made to play on the edge of the piece until a small area around the
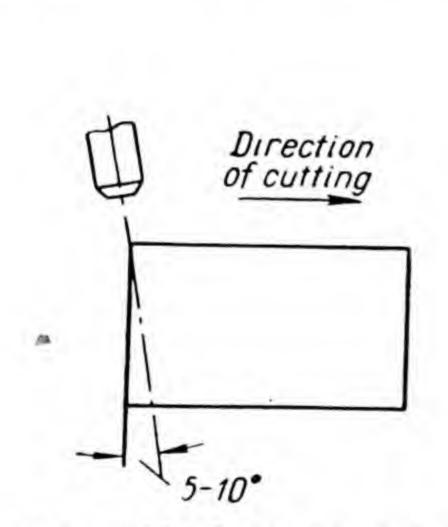
Fig. 165. Starting a cut on heavy gauges
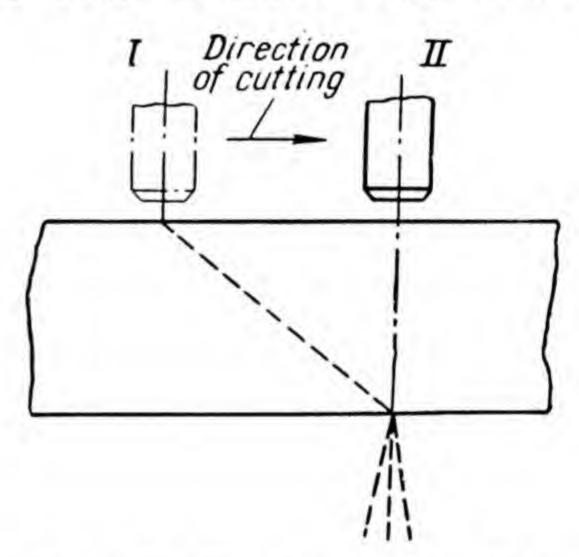
Fig. 166. Starting a cut with a moving torch:
I — beginning; *II* — final position in starting a cut

start of the cut is raised to kindling temperature. In practice, the start of a cut is usually heated until the surface layer of the metal is melted.

In cutting heavy sections, the cutting tip is initially held at an angle of 5 to 10 degrees to the surface of the piece (Fig. 165) to facilitate preheating across the entire thickness of the metal.

In the case of light gauges (up to 50 mm), the tip should be held vertically.

In starting a flame cut in the metal away from the edge, a hole has first to be pierced. In manual cutting and in sections up to 20 mm thick, this may be done with a stationary torch, while the preheating flame is turned off in order to avoid a flashback after the metal has been heated to the required temperature. In practice, the preheating flame is only turned off for a very short time (while a hole is being pierced) and is turned on again to preheat the metal in the immediate path of the cutting jet.

Of late, operators, in cutting steel plate 70 to 100 mm thick, have come to make starting holes by turning on the cutting oxygen jet and moving the torch immediately after this. As a result, the cutting jet gradually cuts through the metal as shown in Fig. 166.

In heavy-gauge plate 50 to 100 mm thick, starting holes are often made with a portable drill or on a drilling machine. In very heavy sections, this purposes can be served by an oxygen lance (see Chapter XXVII).

(3) Tip-to-Metal Distance. The metal is best preheated when the hottest part of the flame is concentrated on the surface of the piece. Practically, this distance is 1.5-2.5 mm.

The tip-to-metal distance, h, for steel up to 100 mm thick can be found by the formula:

$$h = l + 2 \text{ mm}, \qquad \text{(XXIII.1)}$$

where l is the length of the inner cone in mm.

It is essential that the tip be held at an even distance above the surface throughout the cutting operation. In manual cutting this is achieved by mounting the torch on guide wheels, and in machine cutting by levelling up the plate and by using a floating carriage for the torch.

In cutting heavy gauges (over 100 mm), it is a sound practice to maintain the tip-to-metal distance somewhat greater than will be given by formula (XXIII. 1), so that the tip may not be overheated or the orifices clogged by spatter and slag. According to the Autogenous Welding Research Institute, this distance for heavy steel sections cut with low-pressure oxygen may be found by the formula $h = 5 + 0.05\,s$, where s is metal thickness in mm.

(4) The Position and Motion of the Torch during the Cutting Operation. In manual straight-line cuts, the torch should make

an angle of 20 to 30 degrees in the direction opposite to the progress of cutting (Fig. 167). When applied to steel sections up to 20-30 mm thick this technique substantially raises cutting rate.

This advice fully applies to machine straight-line cutting, while in machine shape-cutting the torch should be held at precisely right angles to the surface of the piece.
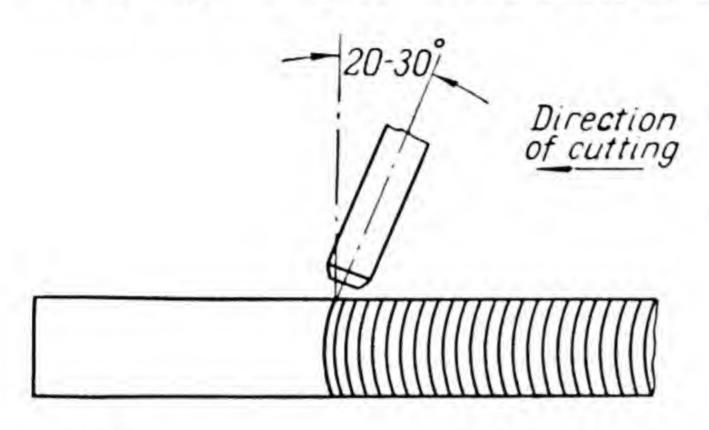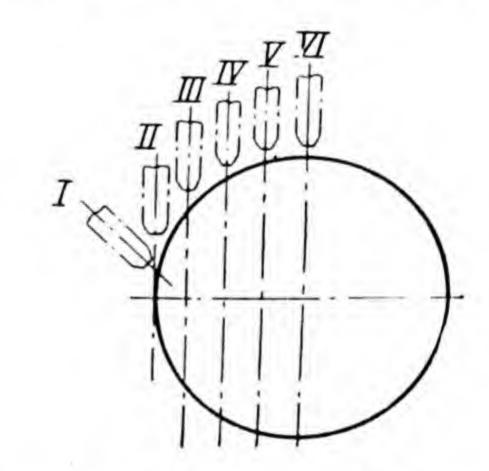
Fig. 167. Position of the torch in cutting steel sheet

When cutting round bar steel, a cut should be started with the preheating flame pointed as shown at *I*, and the cut should

Fig. 168. Position of the torch in cutting round stock

progress with the torch held as shown at *II* through *VI* in Fig. 168.

(5) Cutting Speed. The steady progress of cutting, high-quality cuts, small drag and little slag may only be attained when the torch is advanced at fundamentally the same rate as the metal is oxidised in the immediate path of the cutting jet across the entire depth of the piece. When the torch is advanced too slowly, the edges are melted, while when the rate of torch advance is too high, there may be uncut lengths left and the cutting operation may be interrupted.

When the rate of torch advance is set properly, the shaft of sparks and slag on the bottom side of the cut should be at nearly

right angles to the surface of the piece (Fig. 169a). When the rate of torch advance is too high, the shaft of sparks will make an angle with the metal (Fig. 169b). Yet, operators often use increased rates of torch advance when cutting relatively light gauges, ignoring the drag and rugged edges, as these drawbacks are fully compensated by some increase in cutting rate. In machine straight-line cutting, for example, involving steel sections
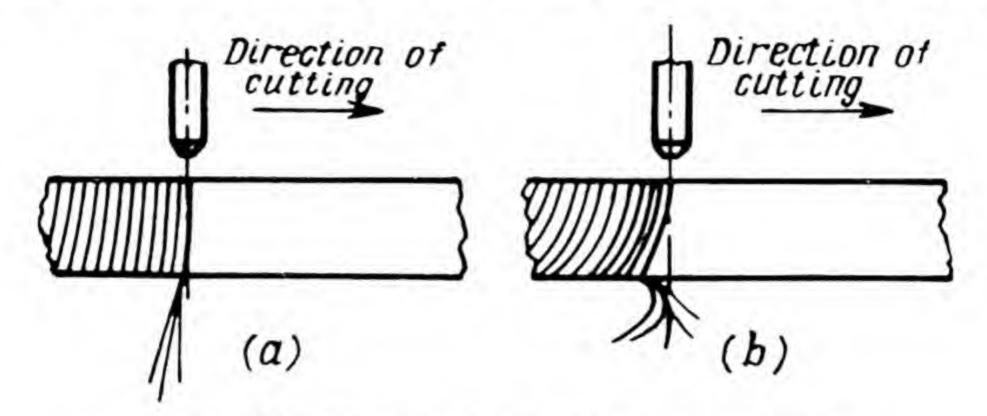
Fig. 169. The shaft of sparks in cutting:
a — normal speed of cutting; b — increased speed of cutting

not more than 25 or 30 mm thick, especially when smooth cut edges are not essential, the drag may be as great as 10 mm and the cutting rate may be raised by as much as 30 or 40 per cent.

Cutting rate depends to a great extent on the purity of cutting oxygen (see Sec. 6, Chapter XX).

The Autogenous Welding Research Institute has derived approximate formulas of cutting speed for ordinary and stepped straight-sided tips:

$$\bar{\omega} = \frac{1,000}{\tau} \text{ mm/min} \tag{XXIII.2}$$

or

$$\bar{\omega} = \frac{40,000}{50 + s} \text{ mm/min,} \tag{XXIII.3}$$

where $\tau = 1.25 + 0.025\,s$ is the cutting time per linear metre of the cut in minutes, and s is metal thickness in mm.

The cutting speed found from the two empirical formulas, however, cannot ensure cuts of high quality. Indeed, this speed is only suitable for scrap cutting and demolition. To obtain high-quality cuts the speed found by the two relationships should be decreased by 10 per cent when rough cutting, by 20 per cent when cutting shapes with machining allowances, by 30 per cent in machine straight-line cutting without any subsequent machining, and by 40 per cent in the case of precision machine shape

cutting. In manual cutting the speeds found by formulas (XXIII. 2) and (XXIII. 3) should be reduced by an average of 20 per cent.

Table 48 gives suggested cutting rates, V_c (m/min), for machine cutting.

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Table 48

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Suggested Speeds for Machine Gas Cutting of Steel

Type of cutting	Steel thickness, mm										
	5	10	20	30	50	80	100	150	200	250	300
Scrap cutting	730	670	570	500	400	310	270	200	160	140	120
Straight-line rough cutting	660	600	520	450	360	280	240	180	150	120	110
Shape cutting with machining allowances .	590	540	460	400	320	250	210	160	130	110	90
Straight-line finish cutting	510	470	400	350	280	220	190	140	110	100	80
Shape finish cutting . .	440	400	340	300	240	190	160	120	100	80	70

Note: The tabulated data apply to cutting with oxygen 98.5-99 per cent pure.

According to the Autogenous Welding Research Institute, the speed of machine straight-line cutting in the case of steel up to 30 mm thick may be doubled or trebled by tilting the tip at an angle of 45° in the direction opposite to the progress of cutting.

When so tilted, the cutting oxygen jet forces the slag away from the forward edge being cut and speeds up the oxidation of the metal. It is believed that cutting speed may also be raised by heating the cutting oxygen jet, for which purpose high-speed torches have their cutting oxygen orifices placed above the preheating flames. To obtain smooth cut edges at high cutting speeds, torches are usually fitted with additional cutting orifices whose duty is to "finish" the kerf faces.

Steel plates may be cut at high speeds in special-purpose machines which have provisions for adjusting cutting speed anywhere between 400 and 2,150 mm/min. Fig. 170 compares speeds for ordinary and high-rate cutting operations.

(6) Heat Input from the Preheating Flame. As expressed in terms of fuel-gas flow, the required heat input from the preheating flame varies with the thickness of the metal to be cut. The principal requirement for the preheating flame is that it should raise the metal quickly to kindling temperature in the beginning

and maintain the metal in the path of the cutting jet at that point throughout the cutting operation. Comparative data on the fuel-gas flow for various metal thicknesses are presented in Table 49.

Fig. 170. Ordinary and high cutting rates related to metal thickness:
1 — high-quality cutting;
2 — rough cutting
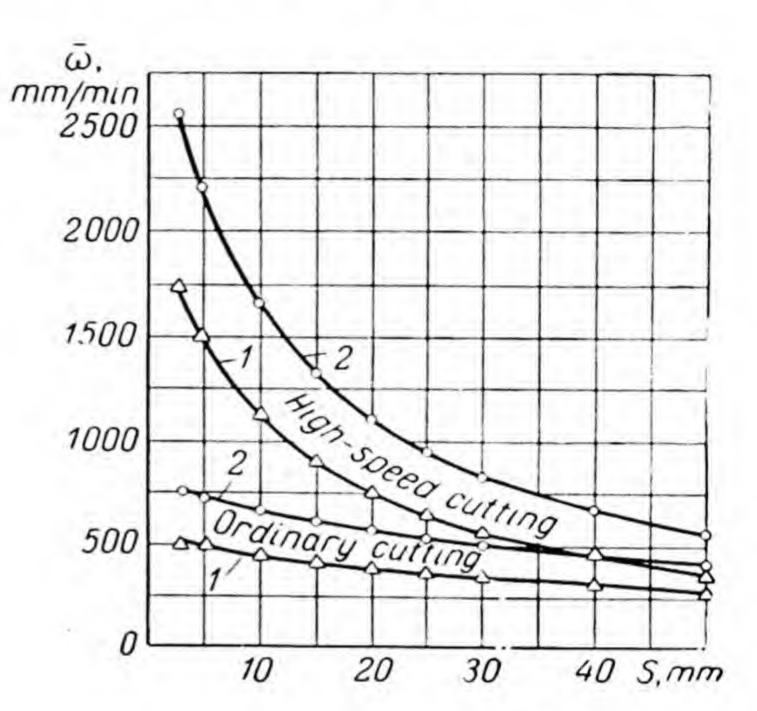
Table 49

Fuel-gas Flow for Flame Cutting

Torch	Fuel gas and Its flow rate	Thickness of steel, mm						
		5	25	50	100	200	250	300
Type PP	Acetylene, m³/hr	0.8	0.9	1.0	1.0	1.1	—	1.2
Type PM	Ditto	0.45	0.6	0.65	0.7	0.9	1.0	—
Type P3P-55	Commercial methane, m³/hr . .	0.8	1.1	1.2	1.4	1.6	—	1.9
Ditto	Coke-oven gas, m³/hr	1.3	1.8	2.0	2.25	2.25	—	3.0
Type K-51	Kerosene, g/hr .	700	800	900	1,100	1,300	—	—

(7) Cutting Oxygen Pressure. Cutting oxygen pressure varies with metal thickness, the shape of the cutting orifice, and oxygen purity. When oxygen pressure is insufficient, the progress of cutting is slowed down, slag accumulates on the bottom side of the cut and in the kerf, and the oxygen jet fails to cut through the entire depth of the piece. When oxygen pressure is too high, too much oxygen is consumed, the metal in the path of the cut-

ting jet is cooled and the progress of cutting is likewise slowed down or hampered altogether. In addition, the cutting jet diverges, and the cut edges appear rough and ragged.

Optimum pressure for cutting oxygen in the case of steel 5 to 300 mm thick is suggested in Table 50.

Table 50

Cutting Oxygen Pressure in atm (gauge) for Flame Cutting of Steel

Torch	Thickness of steel, mm							
	5	10	25	50	100	200	250	300
Type PP	3	—	4	6	8	11	—	14
Type PM	3.5	4.5	4.5	6.5	10.5	—	12.0	—
Type P3P-55	3	—	—	7-9	5-11	8-11	10	12
Type K-51	4	—	5	7	9	11	—	—

(8) Oxygen Consumption. The aggregate oxygen consumption (Table 51) is the sum of the oxygen consumed in the preheating flame and in the cutting jet, or:

$$V_{tot\,ox} = V_{ph\,ox} + V_{cut\,ox}, \qquad (XXIII.4)$$

where $V_{tot\,ox}$ = total oxygen consumption in lit/hr;
 $V_{ph\,ox}$ = oxygen consumption in the preheating flame, lit/hr;
 $V_{cut\,ox}$ = oxygen consumption in the cutting jet, lit/hr.

The oxygen consumption in the preheating flame can be determined on the basis of the fuel-gas consumption and the working

Table 51

Total Oxygen Consumption in Flame Cutting of Steel

Torch	Thickness of steel, mm					
	5	25	50	100	200	300
Type PP	2.6	5.2	8.5	18.5	33.5	42.0
Type PM	2.6	4.6-5	7.4-8.3	13.5-14	28-29	—
Type P3P-55 . . .	3.0	—	10.0	15	26	38
Type K-51	5.4	7.6	9.8	20.2	32.6	—

Notes: 1. The tabulated data apply to low-carbon steel and an oxygen purity of at least 99 per cent.
2. The figures for the Type P3P-55 are tentative.

oxygen-to-fuel gas ratios given in Sec. 4, Chapter VIII, or

$$V_{ph\ ox} = \beta_0 V_{fg}, \qquad (XXIII.5)$$

where V_{fg} is the fuel gas consumption in lit/hr.

2. Cutting with Low-pressure Oxygen

Apart from the necessity to use armoured hoses (at pressures over 15 atm gauge), cutting with high-pressure oxygen entails its waste as a considerable proportion of it does not contribute to the oxidation of the metal, while the cutting jet is cooled due to the wiredrawing (throttling) effect.

Oxygen pressure before the tip may be appreciably reduced without affecting the progress of cutting if, at the same time, the cross-section of the tip orifice as well as the oxygen passages in the torch, pressure regulator and hoses are increased.

Tables 52 and 53 give cutting orifice sizes for machine and manual cutting with low-pressure oxygen.

Table 52

Basic Tip Dimensions for Machine Flame Cutting

(Earlier and Recent Makes)

Sectional view of one-piece tip	Dimensions, mm		Tip size						
			1	2	3	4	5	6	7
	d_{in} old		1.0	1.2	1.4	1.7	2.0	2.4	2.8
	d_{in} new		1.0	1.3	1.7	2.0	2.4	2.7	3.0
	d_{out} old		1.2	1.4	1.7	2.0	2.5	3.0	3.5
	d_{out} new		1.0	1.3	1.7	2.1	2.6	3.1	3.5
			(1.2)	(1.4)					
	l_{out} old		10	10	10	10	10	10	10
	l_{out} new		0 (10)	0 (10)	0	6	8	9	10
	d_{ph} old		0.7	0.7	0.8	0.8	0.9	1.0	1.0
	d_{ph} new		0.8	0.8	0.8	0.9	0.9	1.0	1.0

The notation in tables 52 and 53 is as follows:

d_{in} = diameter of the orifice throat;
d_{out} = diameter of the orifice outlet;
l_{out} = length of the outlet passage;
d_{ph} = diameter of the preheating flame orifice.

Table 53

Basic Dimensions of Inner Nozzles of Two-piece Tips for Manual Flame Cutting (Earlier and Recent Makes)

Sectional view of nozzle	Dimensions, mm		Nozzle sizes					
			1	2	3	4	5	6
	d_{in} { old		1.0	1.4	1.6	2.3	3.0	—
	{ new		1.3	1.7	2.0	2.5	3.0	3.5
	d_{out} { old		1.3	1.7	2.0	2.8	3.4	—
	{ new		1.3	1.7	2.0	2.7	3.3	4.0
						(2.8)	(3.4)	
	l_{out} { old		10	10	10	10	10	—
	{ new		0	0	0	8 (10)	10	12

The dimensions are given solely for the sake of comparison. In drilling out existing tips, the numerals given in brackets should be adhered to.

Tables 54 and 55 suggest cutting oxygen pressures and tip sizes as functions of steel thickness for machine and manual flame cutting, respectively. The tabulated data also apply to Type K-51 oxy-benz torches having cutting oxygen orifices of increased diameter.

Table 54

Tip Sizes and Oxygen Pressure in Regulator

Thickness of steel, mm	Cutting at top speed		Quality cutting	
	Tip size	Oxygen pressure, atm (gauge)	Tip size	Oxygen pressure, atm (gauge)
5-10	1	2.5-4.5	2	1.5-3
10-25	2	2.5-4.5	3	1.5-3
25-50	3	2.5-4.5	4	1.5-3
50-90	4	3-5.5	5	2.4
90-150	5	4-6.5	6	3-5.5
150-220	6	5.5-8	7	4.5-7
220-300	7	7-9.5	—	—

In the case of drilling out the orifices in machine and manual torch tips, it is necessary also to drill out the outlet orifice in the injector to 0.9 mm so that there may be enough oxygen in the

Table 55

Tip Sizes and Oxygen Pressure in Regulator

Steel thickness, mm . .	5-20	20-40	40-70	70-120	120-220	220-300
Tip size . . . . . , . .	1	2	3	4	5	6
Oxygen pressure in regulator, atm (gauge) .	2-4	2-4	2.5-4.5	3-5	4.5-6.5	—

preheating flame at the reduced pressure. When manually cutting steel over 200 mm thick, it is a good plan to fabricate an additional Size 6 inner nozzle to Table 53, keeping its remaining dimensions within the limits given for the Size 5 tip.

To meet the requirements of cutting with low-pressure oxygen, there have also been developed a cylinder pressure regulator and a cylinder valve with increased orifices and passages so as to reduce losses in gas pressure.

When employed in conjunction with tips and oxygen hoses of increased diameters, low-pressure oxygen cutting may be carried out at normal speeds (comparable with that of normal-pressure oxygen) and with a fairly high quality of cuts. In cutting steel up to 50 mm thick with low-pressure oxygen, the pressure of the cutting oxygen before the tip should not exceed 4 or 4.5 atm (gauge), because higher pressures do not add to cutting speed, while oxygen consumption increases.

In cutting steel up to 100 mm thick at one and the same speed, the consumption of cutting oxygen is somewhat greater at the lower than at the higher pressure. In the case of heavier sections, the oxygen consumption is roughly the same in both cases.

In cutting steel 100 to 300 mm thick, the maximum cutting speed is attained with an oxygen pressure (before the tip) of 5 to 8 atm (gauge), while cuts of the highest quality are obtained with a pressure of 3 to 5 atm (gauge).

3. Light-gauge Steel Cutting

The flame cutting of steel above 3 or 4 mm in thickness presents no difficulties. In the case of lighter gauges, the edges are severely fused and the metal is overheated by the preheating flame. For this reason, it is more advantageous to sever light sections with shears rather than by flame cutting. Yet, in some cases steel sheet 2 or 3 mm thick has to be flame-cut. The width of the heat-affected zone can be reduced and the overheating of the edges limited or eliminated altogether by means of tips in

which the preheating flame (or flames) and the cutting oxygen jet are arranged in line with the cut (see Fig. 130 and Sec. 3, Chapter XX). The cutting speed in such a case should be as high as possible and the output of the preheating flame reduced to a minimum.

4. Stack Flame Cutting

In addition to cuts made through single thickness of material, flame cutting may sometimes be employed for cutting through several thicknesses at a time. This technique is termed "stack flame cutting". It is generally applied to sheet steel from 1 or 1.5 mm upwards in thickness, with as many as 25 to 50 thicknesses in a stack (depending on the individual thickness of the metal to be cut). The cuts thus made are of fairly good quality (Fig. 171).
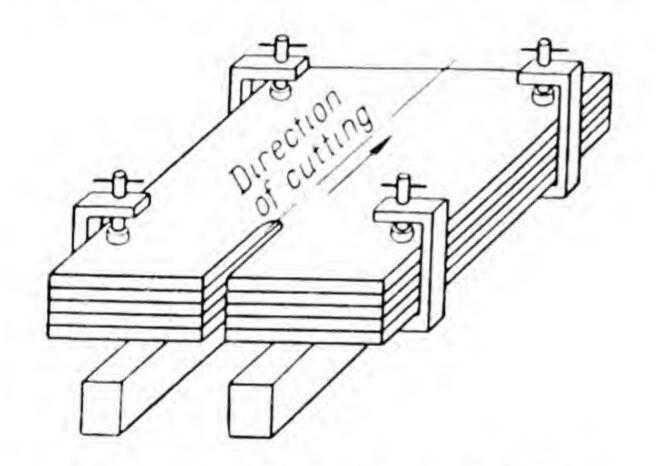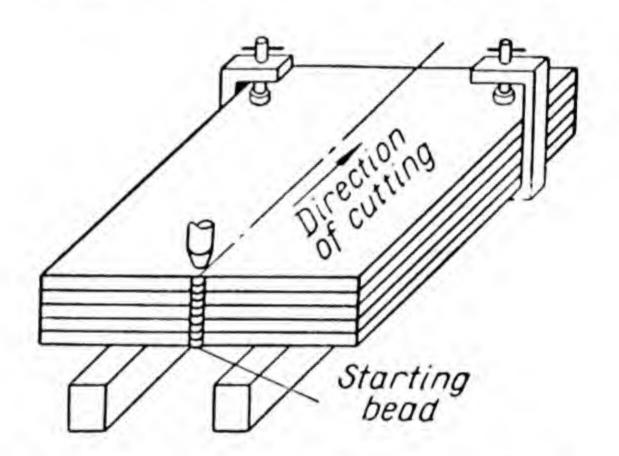

Fig. 171. Stack flame cutting

Fig. 172. A stack of plates held together by cramps, with a bead deposited facilitate cut starting

Stack flame cutting may use oxygen under either normal or reduced pressure.

When normal oxygen pressure is employed, the plates in the stack should be clamped together as tightly as possible, which is accomplished by means of cramps, and the thickness of the individual plates should not exceed 8 or 10 mm, as heavier plates are difficult to clamp tight.

In cutting with low-pressure oxygen, the plates in the stack may be separated by a distance of 3.5 to 4 mm, and the thickness of the individual plates may be up to 30 mm. To avoid likely warpage, it is desirable to clamp the "loose" stack at least at the far end (Fig. 172).
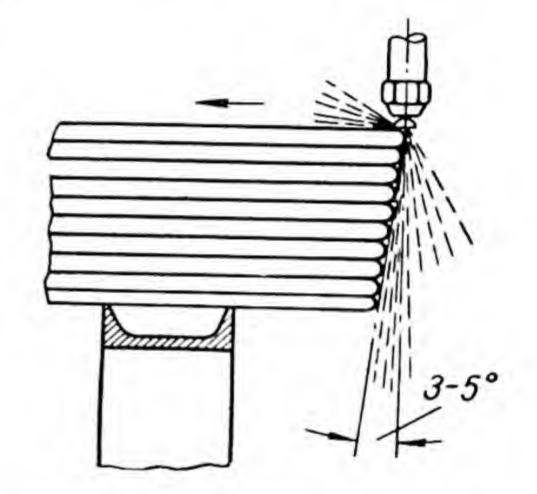
Irrespective of the oxygen pressure employed, the plates in the stack should be flat and cleaned of scale and dirt, with the ed-

ges in alignment where the cut is to be started. The start of a cut may be facilitated by placing the top plates with some overhang relative to the lower plates (as shown in Fig. 173) or by depositing a bead of low-carbon steel at the edge (see Fig. 172).

To avoid damage to the top plate in the stack due to heating and buckling and to the bottom plate (especially when metal thickness is 1 to 3 mm) due to the action of slag, use is sometimes made of what are known as "waster plates" which are clamped onto the bottom and top plates.

Oxygen and acetylene pressures for stack flame cutting are adjusted on the basis of the aggregate thickness of the stack and are roughly equal to the pressure which would be used to cut a solid piece of metal of the same thickness.

Fig 173. Plates stacked for an easier start of the cut

In cutting with low-pressure oxygen the oxygen jet issuing from the central orifice has a low velocity, molten slag runs down the kerf slowly and heats the lower plates well. In addition, when the plates in the stack are piled loose, the slag closes the gaps between the plates, thereby helping to direct the oxygen jet and maintain oxidation across the entire thickness of the metal. This is achieved by using lower oxygen pressure before the tip and a straight-sided cutting oxygen tip of increased diameter. Pressures for stack flame cutting with low-pressure oxygen are the same as in Tables 54 and 55.

5. Heavy Flame Cuts

(1) Factors Affecting Heavy Flame Cuts. Heavy flame cuts necessitate the use of high oxygen pressure. High pressure causes the oxygen jet to whirl and spread as it leaves the tip, thereby producing a wider kerf and poor kerf faces. In addition, high pressure adds to the wiredrawing (throttling) effect, the oxygen jet cools more and the progress of cutting is hampered.

24— 413

Another difficulty encountered in heavy flame cutting is that the composition of the metal varies across its thickness. The layers higher in carbon have a lower melting point and will be ignited with oxygen at a higher temperature than those lower in carbon.

The thickness of the piece also affects the temperature gradient through the metal. As a result, the lower layers pick up heat at a slower rate than those at the top; the drag increases, and slag accumulates at the bottom of the cut. This difficulty may be
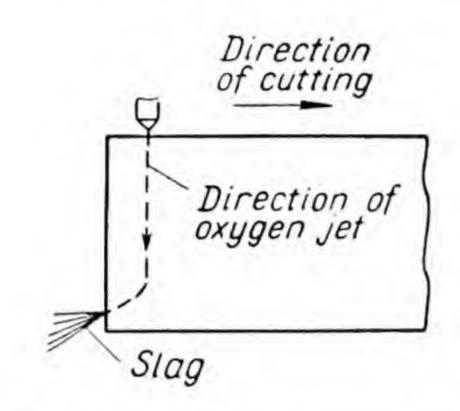

Fig. 174. An indication of an incomplete cut through the workpiece
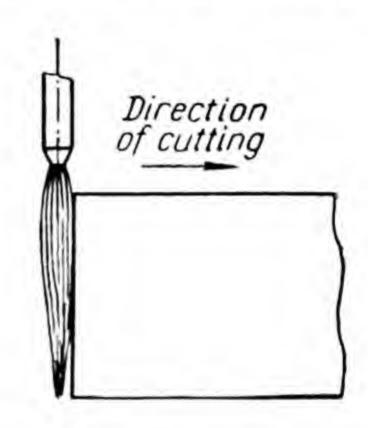

Fig. 175. Checking the cutting oxygen jet for verticality

overcome by preheating the whole workpiece prior to cutting to a temperature of 300 to 500° C in a furnace. Preheating to 250-300° C, or the lower limit is resorted to most often, as it is conducive to high-quality cuts and poses no difficulties to the operator. The point to be watched when the piece is preheated as a whole is that the cut must be started as soon as the piece is withdrawn from a heating furnace. Otherwise, the metal on the outside will cool more than in the inside, and oxidation will proceed at a higher rate in the inside than on the outside. If this is the case, pockets may be produced inside the piece, causing slag to accumulate in the cut and upsetting the progress of cutting.

The position of the torch relative to the surface of the piece and its rate of advance affect the quality of the cut. This is especially true of the start of the cut. When the operator fails to cut through the edge where the kerf is started, the oxygen jet blows slag aside as shown in Fig. 174, the cut is obstructed by the slag, and the progress of cutting is hampered. In many cases this condition causes expensive pieces to be rejected.

The following are a few suggestions towards heavy flame cuts of high quality.

(a) When starting a cut, the cutting tip should be held either at right angles to the surface or tilted through a small angle in the direction opposite to the progress of cutting. In machine cutting, especially when the edge where the cut is to be started is square, it is convenient to check upon the verticality of the cutting jet as follows. The cutting orifice is placed so that its central line is parallel with the vertical edge and is within a few millimetres of it. The preheating flame is ignited, and the cutting oxygen jet is turned on. When viewed against the background of the preheating flame, it is easy to see whether the cutting jet is vertical or not (Fig. 175). After this check, the cutting oxygen jet is cut off, and the torch is brought close to the edge of the piece.

When the torch is tilted, the flame angle should be about 5° to facilitate preheating of the metal and the removal of the resulting oxides at the bottom of the kerf.

(b) After the start of the cut has been raised to kindling temperature and the oxygen jet has been turned on, the torch may be advanced only after the operator is sure that the cut has been started through the entire thickness of the piece.

(c) To reduce drag at the bottom of the cut, which is especially great when heavy sections are severed, it is advisable to slow down the rate of torch advance and to tilt the torch through 10-15° opposite to the direction of cutting towards the end of the cut. This should be done gradually so as not to hamper the progress of cutting.

(2) Cutting Steel up to 600 mm Thick. In the Soviet Union, steel sections over 300 mm thick are cut with Type УPP-600 manual units and ПMP-600 low-oxygen-pressure cutting machines.

A УPP-600 unit consists of a PP-600 torch, a ten-cylinder oxygen manifold, a PK-250 manifold oxygen pressure regulator of high handling capacity, and an acetylene manifold fitted with a cylinder pressure regulator.

The PP-600 torch (Fig. 176) is intended for manual cutting. It has two hoses (instead of three in earlier makes), one each for acetylene supplied under a pressure of 0.1-0.3 atm (gauge) and for oxygen. The torch has a central straight-sided orifice for cutting oxygen and six preheating flames arranged concentrically around the central orifice. Oxygen is supplied to the torch head and tip by a straight length of tubing (about 300 mm long) in order to prevent the cutting jet from spreading as it leaves the tip. The cutting oxygen passages are of increased diameter to

reduce pressure losses. For this reason, the oxygen pressure before the tip is maintained at 2.5 atm (gauge), while its pressure in the pressure regulator is 5 to 7 atm (gauge), with the hose having an inside diameter of 13 mm and a length of 20 metres. The УPP-600 unit has been used to cut steel up to 700 mm thick.
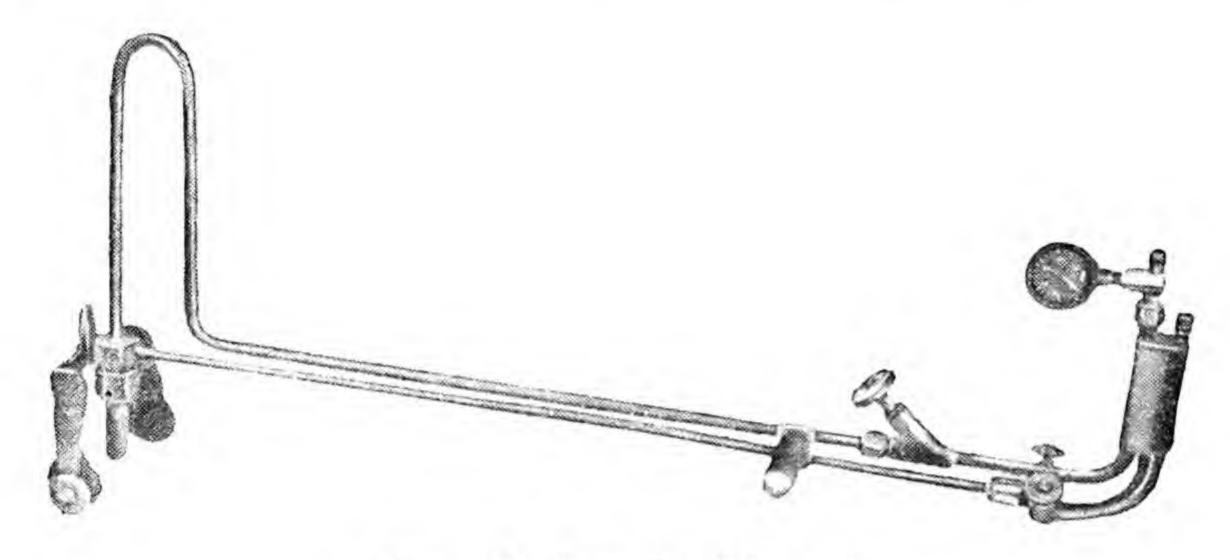

Fig. 176. Type PP-600 torch

The ПMP-600 unit is of the portable type and consists of a Type УXT-52 tractor, an upright, and a PM-600 machine torch.

(3) Cutting Steel over 600 mm Thick. This purpose is served by a torch using an expanding tip (see Fig. 134). This torch, developed by K. K. Khrenov, Corresponding Member of the Soviet Academy of Sciences, and M. M. Bort, an engineer, reduces gas pressure losses and makes it possible to attain the maximum cutting speed at a relatively low oxygen pressure of about 2.5-3.5 atm (gauge). The cutting oxygen jet issues from the tip at a supersonic speed which goes a long way towards stepping up oxidation and improving the removal of the resultant oxides from the kerf.

Fig. 177 shows the Type P-100 torch developed at the Kiev Polytechnical Institute. This torch has a powerful system of twelve preheating flames, concentrically arranged, maintaining the rate of acetylene flow at up to 6,000 litres per hour. It uses a very long oxygen line of increased diameter, interchangeable expanding-orifice tips and water cooling. The torch draws its oxygen and acetylene from two manifolds. The pressure of preheating oxygen is about 5 atm (gauge) and that of acetylene is 1.5 atm (gauge), with the result that the preheating flame has a long flame envelope — an obvious asset where heavy cuts are involved. Every

P-100 torch comes complete with five interchangeable cutting tips, preheating tips and injectors. About the same features are embodied in the Type УБТ-1200 unit intended for cutting steel 500 to 1,200 mm thick.

6. Structural Steel Cutting

This type of steel includes plain medium-carbon steels and steels containing some alloying elements (usually in insignificant quantities).

For the most part, the predominant constituent in structural steels is pearlite or pearlite-sorbite. In a few cases, especially when a steel contains tungsten and molybdenum (steels 18ХНВА and 18ХНМА), martensite is formed even if the steel is allowed to cool in air.

The cutting of structural steels poses no special problems. The cutting procedure for them is the same as for plain low-carbon steels, except for steels high in chromium or silicon. The latter, when being cut, produce refractory oxides which hamper slag removal from the kerf, and the kerf faces are covered by slag.

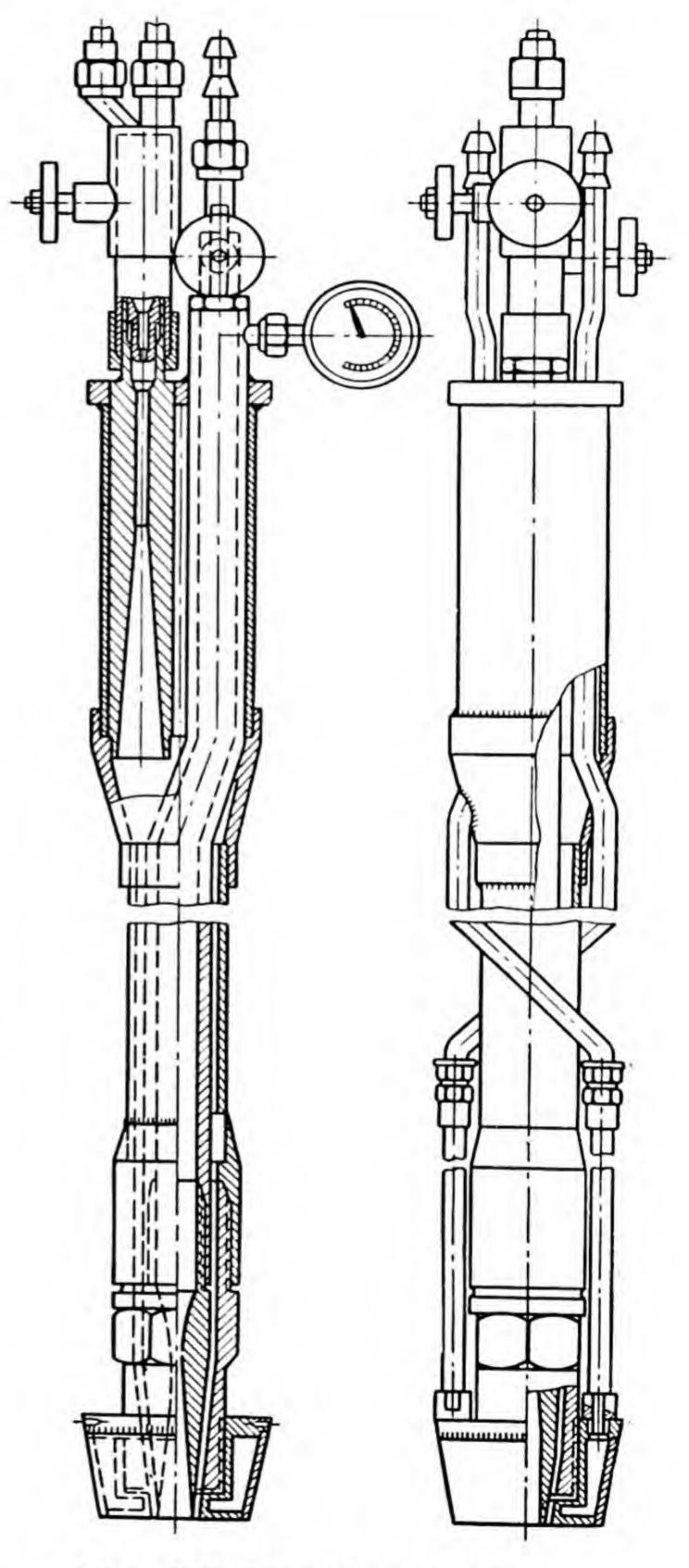The point that should be watched in cutting structural steels is that the cut edges may be hardened, which fact makes difficult their subsequent machining and may result in cracking. These two effects should be attributed to several factors.

Fig. 177. Type P-100 torch

among which are the high temperature of the cut edges, very close to the melting point of steel; the absence of any heat accumulator such as the weld; high rate of heat abstraction by the cutting jet and, as a corollary, the formation of unbalanced quench structures in the heat-affected zone even at the low contents of carbon and alloying elements in the metal.
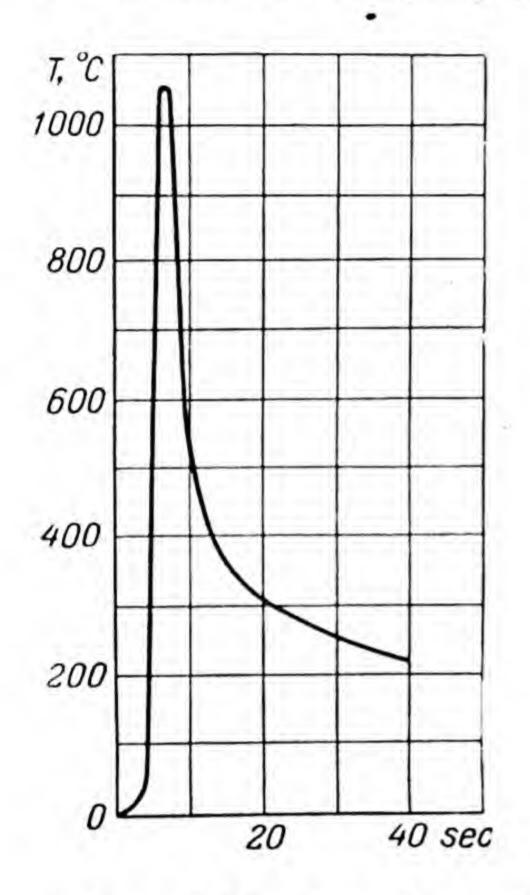
Fig. 178. Rates of heating and cooling in cutting steel 5 mm thick

Fig. 178 gives the rates of heating and cooling for structural steels. The chart is based on the study carried out by G. B. Yevseyev and T. A. Degtyar at the Welding Laboratory of the Bauman Technical School into the thermal cycles of structural steels and the related constitutional changes in the cut edges and the heat-affected zone

As far as the formation of quench structures in steel due to cutting is concerned, of special importance is the rate of cooling in the temperature range where the stability of austenite is at its lowest. For most structural steels this subcritical range extends from 650° C to 450° C. Referring to Fig. 178, the rate of cooling in this range is about 130 degrees C per second. It should be, however, qualified that this rate of cooling is only observed on the surface of the kerf, while within 0.5 to 0.7 mm of the edge the rate of cooling drops to about 40 degrees per second.

(1) Temperature and Hardness Distribution in a Plane Normal to the Cut. An analysis of the thermal cycle (heating and

Fig. 179. Rates of heating and cooling in cutting low-carbon steel specimens measuring $200 \times 200 \times 5$ mm: *a* — arrangement of thermocouples; *b* — variations in temperature with distance from kerf and with time; *c* — temperature gradient across kerf at 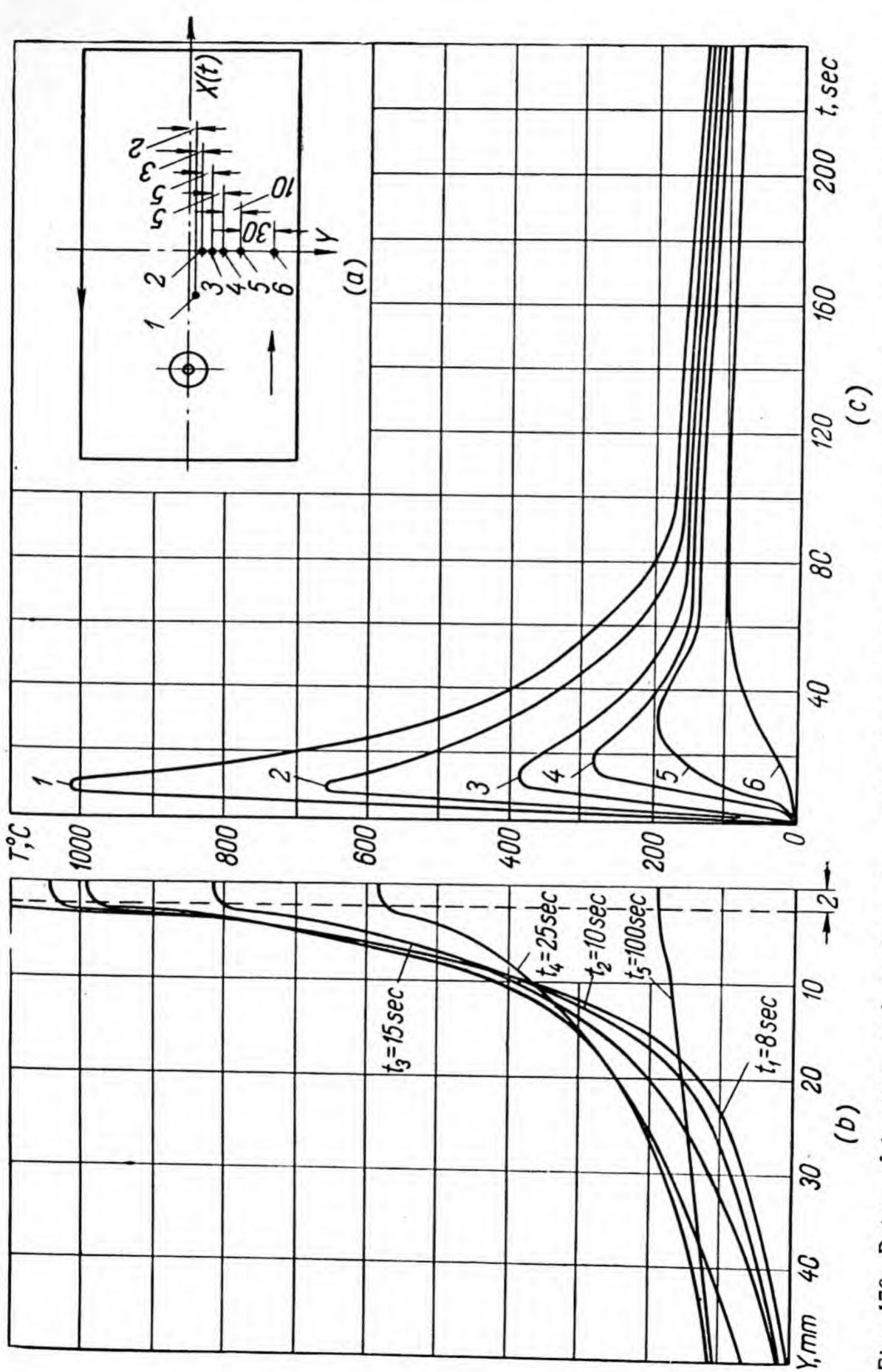different moments of the cutting operation. Curves *1* through *6* have been plotted by means of thermocouples *1* through *6* arranged as shown in "*a*"

cooling) near the cut in steel sheet will show that the maximum temperature of the metal goes progressively down as we move away from the kerf and that the temperature peaks are attained with a time lag. Experimental data (Fig. 179) indicate that for the majority of structural steels 5 mm thick, the recrystallisation temperature of which is anywhere between 700 and 900° C, the zone of constitutional changes is 3 to 5 mm wide.

Within 0.75 mm of the cut edge, the rate of cooling in the subcritical range of temperatures (650-450° C) is about 40 degrees per second, which is sufficient for quench structures to develop in many grades of structural steels.

(a)
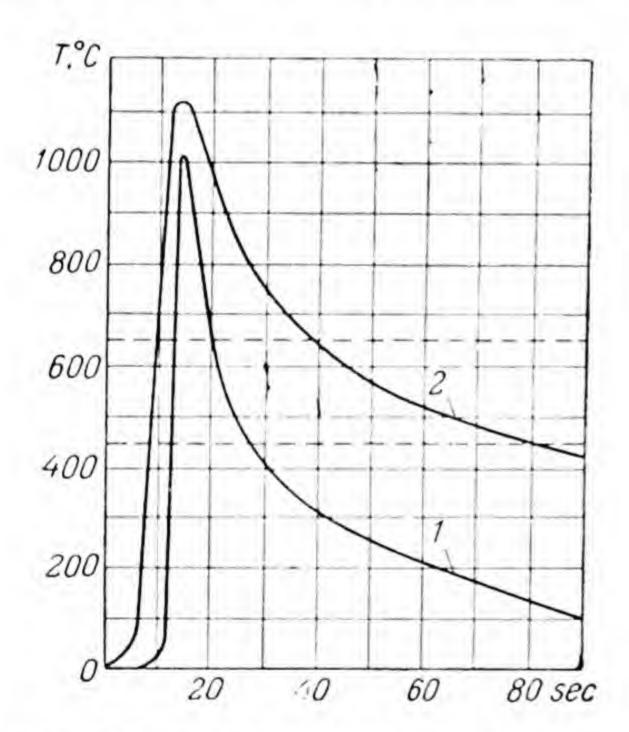
Fig. 180. Thermal cycles in cutting steel 35ХГСА, 5 mm thick:

1 — without concurrent heating;
2 — with concurrent heating

(b)
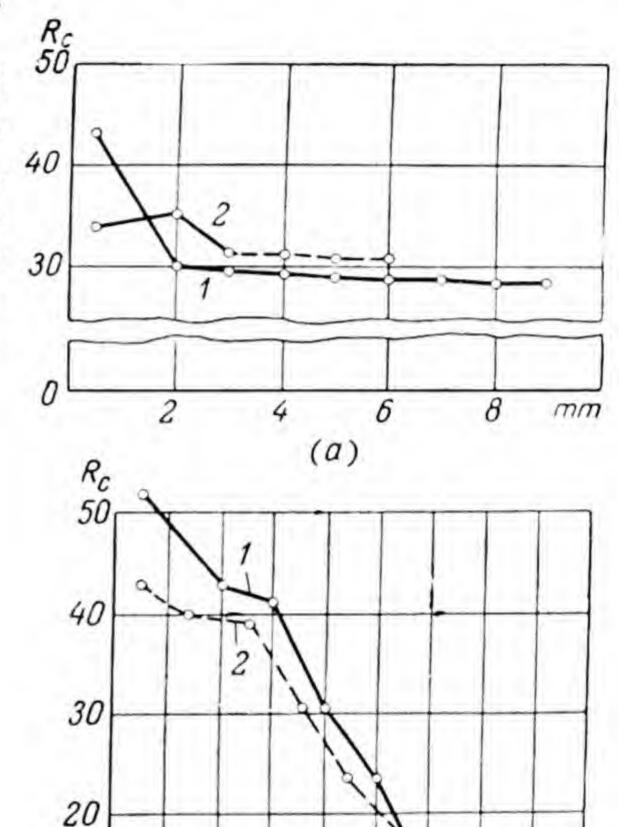
Fig. 181. The hardness of (a) steel 35ХГСА and (b) steel 35ХГСА at different distances from the kerf:

1 — without concurrent heating;
2 — with concurrent heating

This conclusion agrees well with investigators who have studied the hardness gradient in some types of structural steels across the line of cut.

(2) Concurrent Heating in the Cutting of Thin Sheet. The heating of workpieces prior to cutting is a practical proposition

both for pieces of small size, when the cutting operation can be completed within a very short span of time, and for large pieces of substantial mass which cool slowly.

To avoid the hardening of the edges and cracking in sections about 4 to 6 mm thick, it may be advisable to use concurrent

Fig. 182. An attachment for concurrent heating:
1 — cutting oxygen orifice; 2 — torch tip; 3 — concurrent heating orifice; 4 — annular preheating flame orifice

heating with a multiflame torch. As the chart in Fig. 180 shows, concurrent heating with additional flames concentrically arranged around the central orifice reduces the rate of cooling within 0.5 mm of the edge to about 4.5 degrees per second in the temperature range 650°-450° C. This reduction by a factor of eight, as compared with cutting without concurrent heating, is responsible for the lower hardness of the kerf faces and the heat-affected zone. Fig. 181 *a* and *b* shows changes in the hardness of the 20ХГС and 35ХГСА steels with and without concurrent heating. In these experiments, concurrent heating was effected by means of a copper annular attachment with six additional orifices accounting for a total rate of acetylene flow of 900 to 1.200 lit/hr (Fig. 182).

As concurrent heating involves increased gas consumption, it ought to be employed in extreme cases, especially when cracking presents a problem. Concurrent heating is hardly applicable to steel over 6 or 8 mm thick because of the nonuniform distribution of heat across the metal. The latter condition may be avoided, though, through a considerable increase in heat input from the heating attachment or by heating the piece from both sides with special torches — an unattractive proposition in either of the cases

When the cut edges have to be machined, the piece has often to be annealed or normalised in a furnace after cutting. In some cases this purpose may be served by a powerful multiflame torch moved behind the cutting torch.

7. Titanium Flame Cutting

Low specific gravity (4.505), high mechanical strength, resistance to corrosion and relatively high thermal stability make titanium a better material than some structural steels in many cases.

Titanium has a high melting point of 1,680° C and a low thermal conductivity of 0.036 cal/cm per sec per deg. C. However, its coefficient of thermal expansion (8.5×10^{-6} per degree C) is lower than that of austenitic chrome-nickel steels.

At 885° C titanium undergoes allotropic transformation from the alpha- to the beta-phase. In the low-temperature alpha-phase the atoms are in a hexagonal close-packed arrangement, while in the high-temperature beta-phase, the atoms are in a body-centred cubic arrangement. The alpha-phase is more plastic than in the case of some other metals having the hexagonal structure. Under unbalanced conditions of phase changes (quenching), the α'-phase is formed whose lattice is in effect the distorted lattice of the alpha-phase. The α'-phase shows needlelike crystals (like those of martensite in steel) while the alpha-phase consists of globular crystals.

Titanium is a very active element. From the viewpoint of flame cutting which involves heating and oxidation, of special importance is its affinity for oxygen.

Titanium begins to absorb oxygen at 400° C and does so more and more as its temperature rises, when it also absorbs nitrogen and hydrogen.

At below 600° C intensive oxidation of titanium is hampered by the surface oxides and nitrides of titanium. As the temperature rises, the surface oxides and nitrides dissolve in the titanium, and oxygen diffuses in the metal at a high rate. Oxygen readily

dissolves in both the alpha- and beta-phases, forming solid solutions by substitution. Oxygen acts as an efficient stabiliser of the alpha-phase, as it raises and extends the temperature range within which the alpha-phase changes to the beta-phase. When oxygen in titanium runs more than 14.5 per cent, which is above its maximum solubility in the alpha-phase, titanium forms a number of intermediate phases, such as the γ-phase (TiO), the δ-phase (Ti$_3$O$_2$ or Ti$_4$O$_3$), etc. Likewise, nitrogen reacts with the alpha- and beta-phases to form solid solutions by substitution and stabilises the alpha-phase. When it runs high, nitrogen forms intermediate crystal phases, such as the ε-phase (Ti$_3$N) and the δ-phase (TiN).

Hydrogen forms solid solutions in titanium, reacts to produce the hydride TiH$_2$, and stabilises the beta-phase of titanium. As the temperature falls, the solubility of hydrogen in alpha-titanium decreases sharply and is 0.0029 per cent at room temperature.

This reduction in hydrogen solubility is especially marked between 100° and 150° C, and below this temperature the hydride (the gamma-phase) precipitates. In quenching, the hydride may precipitate as dispersed particles, while in slow cooling it may appear as lamellae. The formation of the gamma-phase is accompanied by volumetric expansion and secondary stresses which may result in cracks, especially when the primary stresses have been high.

As distinct from the absorption of oxygen and nitrogen, that of hydrogen is a reversible process. By annealing commercial titanium at 750-800° C in a vacuum of 10^{-3} mm Hg, it is possible to reduce the hydrogen content to 0.002 per cent.

Commercial titanium contains up to 0.5 per cent impurities (max. 0.03 nitrogen, max. 0.15 oxygen, max. 0.15 iron, max. 0.1 magnesium and max. 0.05 per cent silicon), with the result that the strength of commercial titanium is twice that of pure iodide-titanium although its ductility is lower in proportion.

Mechanical Properties of Commercial Titanium

Ultimate tensile strength	45-56 kg/mm^2
Yield strength	11,500 kg/mm^3
Elongation	25 per cent
Reduction of area . . .	55 per cent
Modulus of elasticity .	11,500 kg/mm^3
Hardness	140-200 Brinell

Titanium alloys containing a variety of elements have much higher mechanical properties.

The various titanium alloys are usually classified in terms of the crystal structure at room temperature: all-alpha (HCP), alpha-beta (two-phase) and all-beta (BCC). The presence of either or both of the two phases depends on the content of the elements stabilising the alpha-phase (Al, Sn, O, N, C, etc.) or the beta-phase (Mo, Mn, Cr, Fe, etc.). In many alloys elements from both groups are present.

The authors have investigated the flame cutability of titanium in the Welding Laboratory of the Bauman Technical School. It has been established that the flame cutting of titanium presents no difficulties and the cutting operation proceeds as smoothly and steadily as in the case of plain low-carbon steel. The only distinction is that titanium can be cut at high speed, and oxidation is accompanied by strong incandescence, like that of burning magnesium, in the reaction zone.

In view of the pronounced sensitivity of titanium and its alloys to overheating and to the effect of the oxygen jet, the ordinary torches available on the market are little suitable for titanium sheet cutting as they produce an excessively broad heat-affected zone and severe oxidation of both the titanium and the impurities near the cut edges.

Fairly good cuts in thin titanium sections can be obtained by using a preheating flame, reduced flame output, concentrated heat flux and cutting speeds which are much higher than is considered customary for plain low-carbon steels.

The cut should be started away from the edge, using a waster plate which should be abutting with the titanium plate.

The conditions for the machine straight-line cutting of commercial titanium are summarised in Table 56.

Table 56

Metal thickness, mm	Preheating flame characteristic	Working pressure of cutting oxygen, atm (gauge)	Cutting speed, mm/min
2.5	Annular flame, 300 lit/hr, acetylene . . .	4	2,600-2,800
5.0	Annular flame, 300 lit/hr, acetylene . . .	4.5	1,500-2,600
13.0	Multiple flame	5-6	900

Note: In the case of machine shape cutting, the speed is 10 to 15 per cent lower.

Allotropic transformation in the cut edges and the heat-affected zone takes place as follows. The zone within 0.1 mm of the kerf shows a whitish, poorly etchable needlelike or, sometimes, columnar structure, produced by saturation of the metal with oxygen and nitrogen (Fig. 183a). Then follows the heat-affected zone also showing a needlelike (martensitic) structure (the α'-phase) which results from allotropic transformation from HCP to BCC to the

(a)   (b)
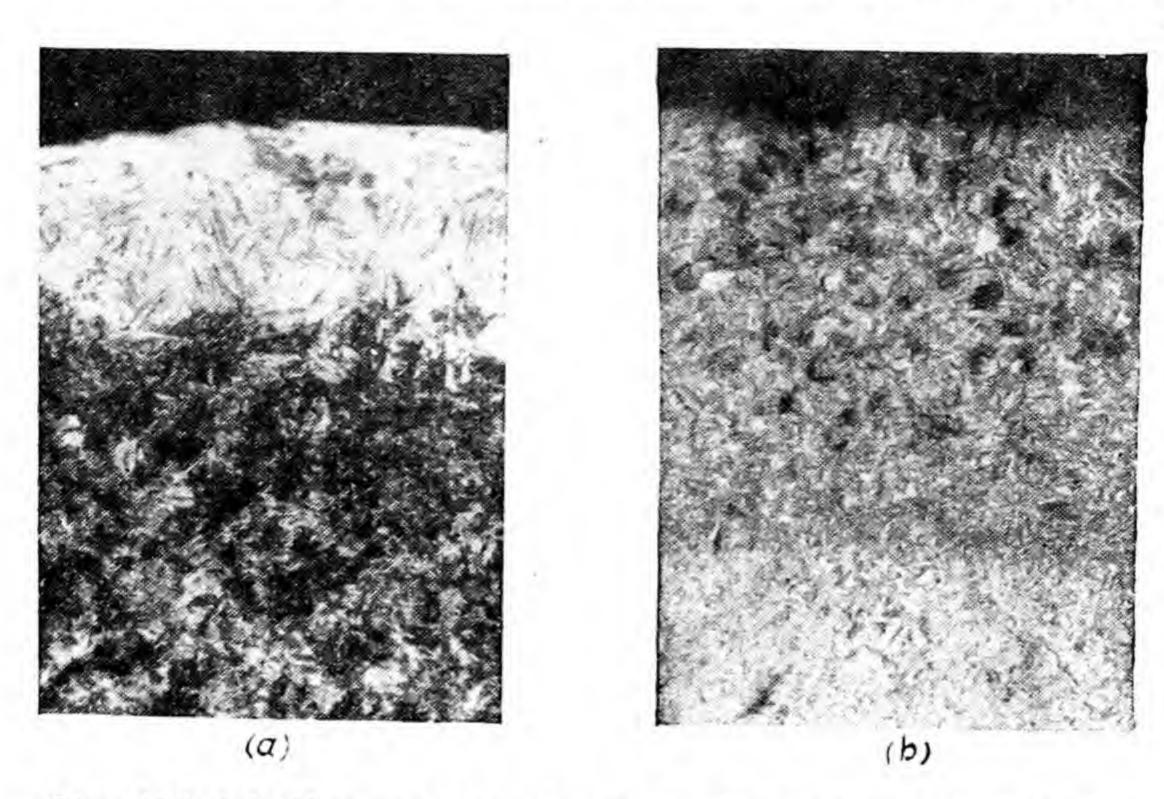
Fig. 183. (a) Photomicrograph of the kerf in commercial titanium, $\times 270$ magnification; (b) photomicrograph of the heat-affected zone in commercial titanium, $\times 120$ magnification

α'-phase during heating and subsequent rapid cooling in the process of cutting. This zone is characterised by the varying size of grains which grow finer as we move away from the kerf. This variation in grain size gives a measure of recrystallisation (Fig. 183b).

As the flame cutting of titanium is an efficient operation and produces a very negligible thermal effect on the metal, it can successfully be employed for both blank-cutting and shape fabrication.

Flame-cut bevels for welding may be ground with an emery wheel to a depth of 0.2 or 0.3 mm, which is sufficient for the removal of the alpha-zone. In the case of critical components, it is essential to remove all of the heat-affected zone, which can be done by machining to a depth of 1 to 1.2 mm.

8. Flame-cut Precision

By precision in flame cutting is meant the agreement in dimensions between the cut component and its drawing and the quality of the kerf.

The dimensional agreement primarily depends on the warpage of the plate due to nonuniform heating and, in the case of machine cutting, on the accuracy with which the torch follows the motion of the tracing head.

The smoothness and squareness of the cut depends on whether the operator uses the proper cutting procedure and technique.

The cutting operation may or may not leave some stock around the contour or along the kerf for subsequent machining. Some stock is usually left when flame-cutting is employed:

1. To fabricate parts to be joined with other parts by welding (the allowance may be anywhere from 0.5 to 3.5 mm, depending on the welding technique to be used).

2. To cut parts which are to be machined or ground with an abrasive wheel (the allowance may be anywhere from 1.5 to 5.5 mm, depending on the machining operation to be used).

3. To remove risers and gates from steel castings to be forged or rolled (the allowance is over 5 mm).

No stock is left for subsequent machining when flame cutting is employed:

(a) to cut parts which are not to be mated with other parts, such as the outside diameter of flanges, the soles of base feet, etc. (the tolerance is up to 5.5 mm);

(b) to cut parts to be lap-joined with other parts by welding, rivetting or bolting (the tolerance of the order of 1.5 to 2.5 mm);

(c) to cut parts which are to be precision-assembled with other parts or to move with respect to them, such as the gears and sprockets of manual drives or low-speed transmissions, ratchet wheels, coarse-lobed cams, etc. (the tolerance is 0.5 to 1.5 mm).

(1) Distortion. The distortion of the plate being cut and of the flame-cut component is due to the appreciable internal deformation set up by local heating. As in welding, the magnitude of internal deformation is governed by the nature of the thermal field, temperature, the temperature gradient both along and across the line of cut, and whether or not the plate is free to expand on heating and contract on cooling.

As the cutting stresses inevitably exceed the yield point of the plate at some places, there always is an amount of residual or locked-up stresses present.

Though main emphasis is usually placed on internal deformation, especially bending deformation, in light gauges it is as important to take into account external deformation.

A number of investigators have established several relationships for external deformation in steel Ст. 3 and 20Г sheet.

Fig. 184 relates the buckling of the plate to the width of the strip cut from it when the metal has fully cooled. The curves are
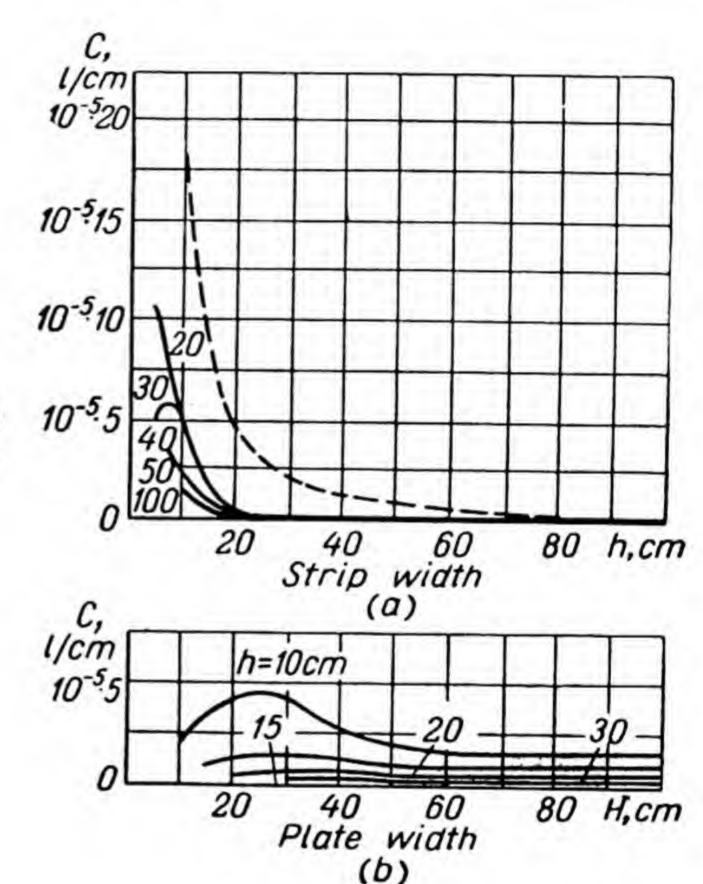
Fig. 184. Buckling in cutting as a function of the width of the part and original plate

plotted for two cases, namely (a) when a strip is cut off the plate, which involves one cut (the dotted line) and (b) when a strip is cut out of the plate, which involves two cuts (the family of solid curves).

The values of buckling, $C = f(h)$, have been obtained analytically on the assumption that the thermal field in oxy-acetylene cutting is produced by two sources of heat operating simultaneously and moving at an equal speed. The two sources are (1) the concentrated source of heat supplied by the inner cone of the flame and the oxidation of the metal with the cutting oxygen ($q_c = 185$ cal/cm per millimetre of metal thickness) and (2) the distributed source of heat supplied by the flame envelope ($q_d = 63$ cal/cm per millimetre of metal thickness). The material used was steel 20Г with a yield strength of 35 kg/mm².

In the calculation of buckling it was assumed that:

(1) every next cut would be started when the part had fully cooled after the previous cut;

(2) both the plate and the part would be free to expand on heating and to contract on cooling;

(3) the cut would be made to a pattern laid out on the piece;

(4) the length of the piece was four times or more its width.

Referring to Fig. 184, the buckling of the plate decreases as the width of the strip both cut off and out of the plate increases. This relationship, however, holds for strips over 10 cm wide. For narrower strips, another relationship exists.

It also follows that buckling will be greater when a strip is cut off the plate (the dotted curve) rather than when it is cut out of the plate (the family of solid curves). It should also be taken into account that the buckling of the strip cut off the plate by two cuts also depends on the width, H, of the remaining plate, being less noticeable as the width H increases.

On the basis of the charts in Fig. 184 it is possible to describe the deflection of flame-cut strips as

$$f = \frac{Cl^2}{8},$$ (XXIII.6)

where l = length of the strip to be cut;

C = buckling of the strip, which is a function of its width, h, when the strip is cut off the plate, or as a function of the width, H, of the original plate when the strip is cut out of the plate.

As the buckling of a flame-cut part increases as the square of its length, narrow and long parts which are to be fabricated to close tolerances should be cut from wider plates.

Fig. 185 shows an alignment chart for determining the deflection of flame-cut strips and remaining plate, based on experimental data
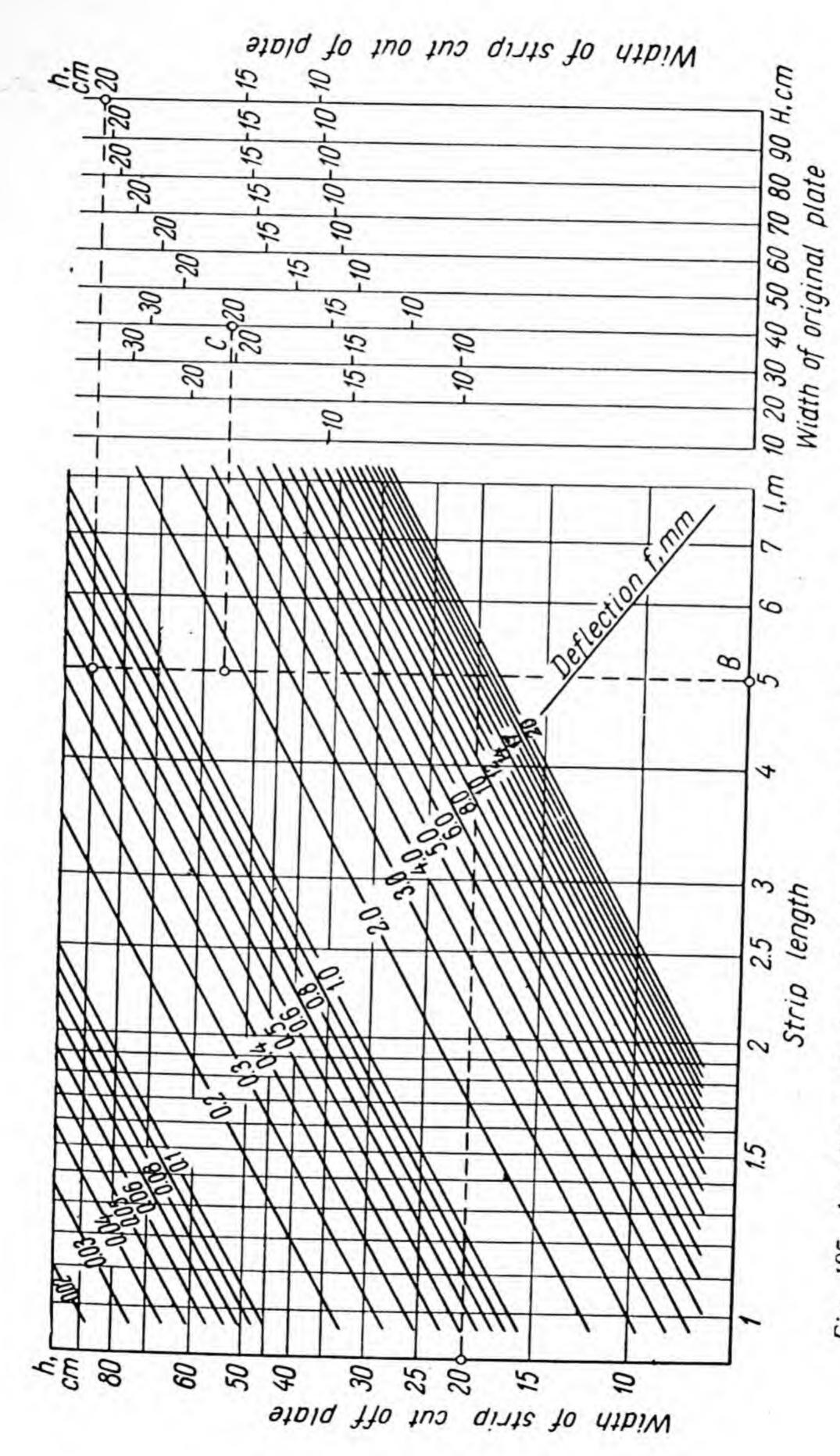
The vertical axis on the left gives the width of strips cut off the plate (the one-kerf case). The vertical lines on the right, each corresponding to a definite width of the original plate, indicate the width of strips cut out of the plate (the two-kerf cases). The horizontal axis gives the length of the strips or the remaining plate. The inclined lines give the deflection of the flame-cut strips.

The alignment chart is extremely simple to use. Here are two examples of its use.

Example 1. Find the deflection of a strip 5 m long and 20 cm wide, cut off a plate with one kerf.

Find $h = 20$ cm on the left-hand vertical axis and draw a horizontal line from the point found. Draw a vertical line from the 5-metre point on the

Fig. 185. An alignment chart for determining the deflection of cut strips and remaining material

horizontal axis. The point of intersection of the two lines will coincide with an inclined line on the alignment chart to give a deflection of about 17 mm.

Example 2. Find the deflection of a similar strip cut with two kerfs out of a plate 90 cm wide.

Find the respective vertical axis on the right ($H = 90$ cm) and on it the strip width $h = 20$ cm. Draw a horizontal line from this point until it crosses the vertical line raised from point B corresponding to the length of the strip, $l = 5$m. The intersection of the two mutually perpendicular lines, superposed on an inclined line, will give a deflection of approximately 0.58 mm.

This alignment chart is only valid for straight strips and plates of constant width along their length. It is possible, however, to construct a diagram of change in buckling C along the length
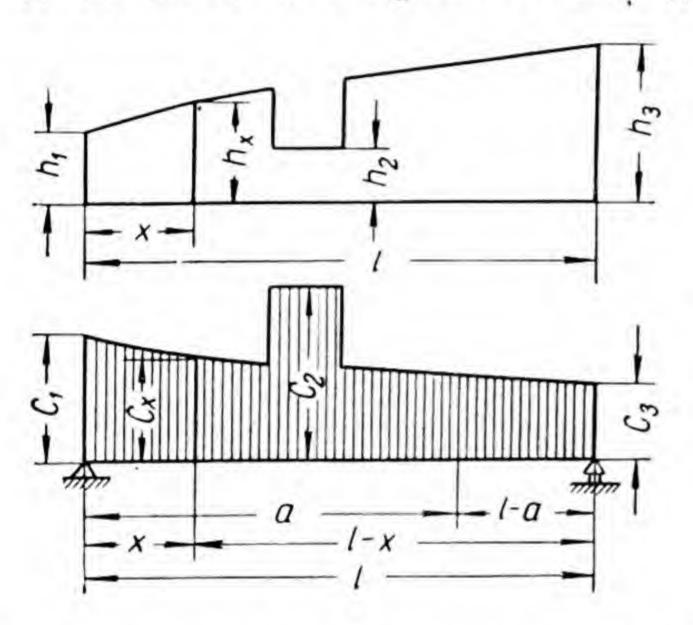
Fig. 186. Determination of the deflection of a strip varying in width

of the piece on the basis of the charts in Fig. 184, assuming that the change in buckling is proportional to that in the width of the piece. This involves the knowledge of the changes in the width of the piece along its length and of whether the piece is cut off or cut of a plate (with one kerf or two).

When a strip of variable width is cut off the plate with one kerf, the relationship $C=f(h)$, shown by the dotted line in Fig. 184a, holds. When a strip of variable width is cut off the plate with two kerfs, both the change in its width and the width of the original plate have to be known. With the width H known, buckling, C, for the length l can be found from the family of solid curves in Fig. 184 a and b.

With buckling, C, for the length l known (Fig. 186), it is possible to determine the deflection of the strip at any cross-section. According to structural mechanics, the deflection of a cross-section is determined as the bending moment under an imaginary distributed load C, where C is buckling.

The cutting technique used has a direct bearing on the final dimensions of the flame-cut piece. In machine cutting from a

template, the dimensional accuracy is appreciably lower than in manual cutting to a pattern laid out on the piece, when the operator follows the profile irrespective of warpage.
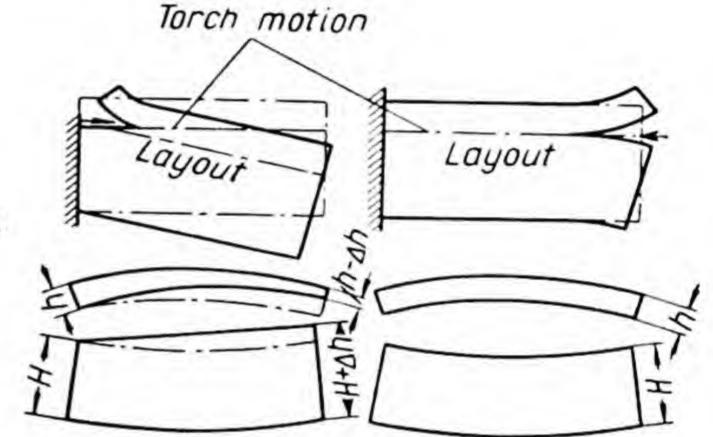
The direction of machine cutting likewise affects the dimensional accuracy of the finished piece.

Fig. 187 illustrates two cases of machine cutting. In one of them a strip is cut off a plate away from the fixed to the free end, and in the other towards the fixed end.

Referring to Fig. 187, in the former case, *a*, both the strip and the original plate undergo a bending deformation due to nonuniform heating across the depth of the material.

Fig. 187. The effect of the direction of cutting on cutting precision

As it bends, the remaining plate causes the whole of the plate to turn about the fixed end, and the path of the torch will not coincide with the laid-out line of cut. As a result, both the strip and the remaining part of the plate will have dimensions other than specified.

In the latter case, *b*, the plate will likewise bend due to nonuniform heating, but the path of the torch will coincide with the laid-out line of cut, and there will be no deviation from the specified width both in the strip and in the plate.

To sum up, when, in machine cutting, the plate cannot, for one reason or another, be clamped all the way round its perimeter and is instead fixed at one end, the cut should be made away from the free towards the fixed end.

In flame cutting from a template with the plate not clamped, cutting deformation will cause the plate to move on its supports in the direction of least friction. The resultant components will, therefore, suffer from inaccurate dimensions as compared with the drawing. This drawback may be eliminated by using suitable cutting sequences. One of them is known in the Soviet Union as the third-cut method.

25*

The method (Fig. 188) consists in that the scrap is first trimmed off the plate in two subsequent cuts, and then the strips are cut off. Starting with the third strip, each strip has two cut sides.

Another method for controlling distortion in rectangular pieces is to perform cutting with two manual or machine torches simultaneously about the neutral axis of the part. This technique
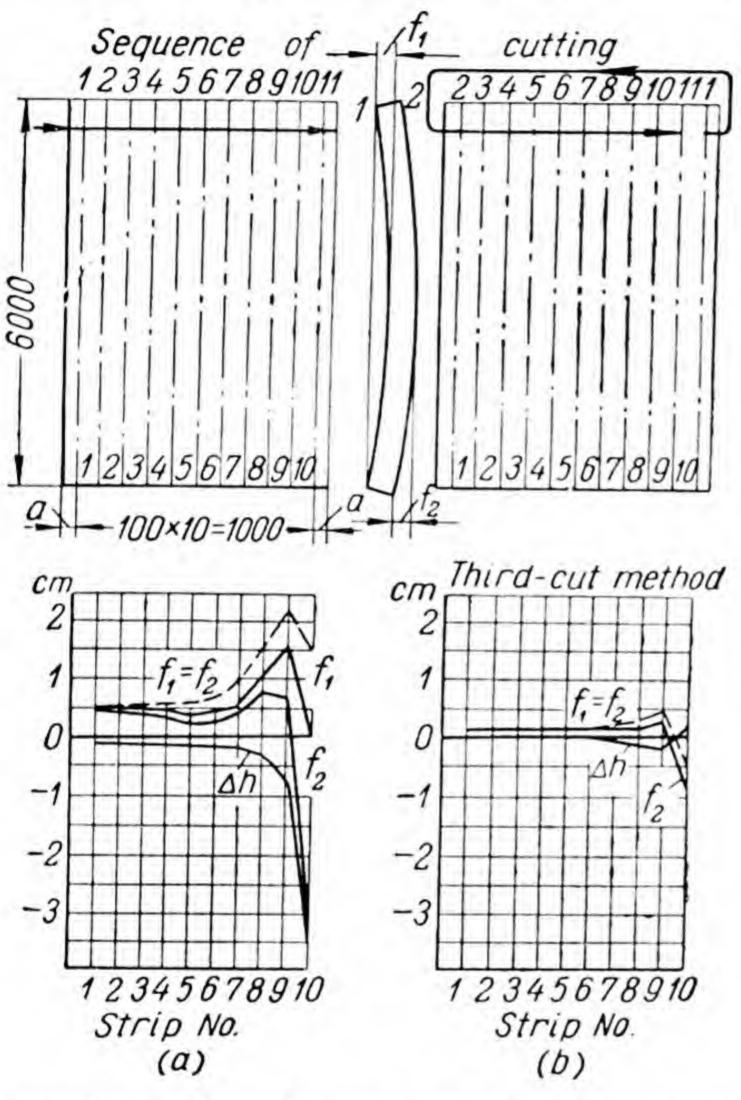
Fig. 188. The effect of cutting sequence on buckling

equalises the forces about this axis, which in turn reduces their physical effect on the part to zero. As a further refinement to the twin-torch technique, some scrap may be left between two adjacent strips 30 to 40 mm wide, to take in such deformation as may be brought about by the cutting operation (Fig. 189).

In the case of cutting from a template, the effect of buckling may be reduced by giving the template some counter-deflection, usually equal to the expected or calculated buckling of the part due to cutting, but in the opposite direction. Fig. 190a illustrates the use of counter-deflection in a template in the case of a rectangular piece cut along one of its sides. The template is given a counter-deflection f_c which is equal to the deflection of the remain-

ing part of the plate, f_h, but with the opposite sign: $f_c = -f_h$. The value of f_h can be found from the alignment chart in Fig. 185 as a function of the width of the part, h. Instead of moving in a straight line, the torch will have to be advanced along a curve (shown by a dotted line in Fig. 190a), which follows the counter-deflected side of the template. Upon cooling, the cut edge will be straight, but the opposite edge will buckle, and the width of the part will not be constant (Fig. 190).
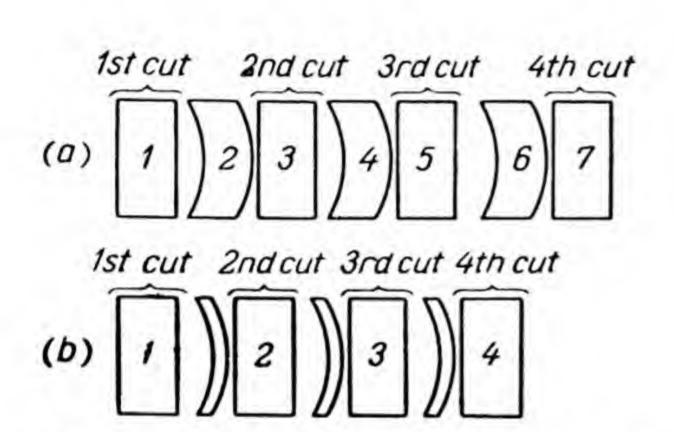
Fig. 189. Strips cut with two torches (the sequence of strip cutting): a — buckling of cut strips; b — strips cut with some scrap left
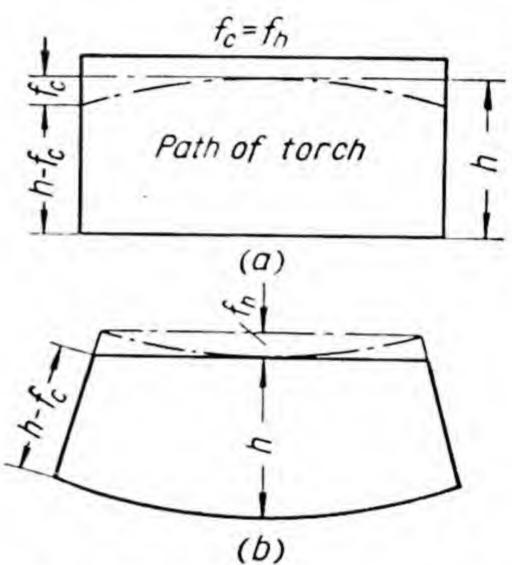
Fig. 190. Counter-deflection of a template on one side to control buckling of a rectangular workpiece: a — template with a counter-deflected edge; b — workpiece

When both edges of a strip have to be straight, counter-deflection should be given to both long sides of the template. The amount of counter-deflection can likewise be found from the alignment chart in Fig. 185.

Further measures of controlling warpage include:

(a) water cooling of the cut edges;

(b) high cutting speeds to reduce metal heating;

(c) skip-cutting in which the cut is made to skip at intervals of 600 to 800 mm, leaving a series of uncut sections along the line of cut, each 6 to 10 mm long; these uncut ligaments hold the material in line until cooled, after which they are cut through to separate the parts.

As a general remark, warpage due to cutting is a very complex problem, and the investigations carried out in this field to date have only tackled isolated cases, particular shapes, some sizes and some thicknesses.

(2) Kerf Width and Quality. The width and quality of the cut
vary with the thickness of the metal and with the cutting method
(manual or machine) employed. In the heavier sections the cut
is wider and less smooth than in light gauges. The cut is likewise
less square in heavy plate, being wider towards the bottom of the
kerf. Furthermore, manual cuts are always wider and less smooth
than machine cuts, as the rate of torch advance is less uniform in
manual than in machine cutting (Table 57).

Table 57

**Cut Width and Surface Irregularities for Steel
5 to 300 mm Thick**

Cutting	Metal thick-ness, mm	Average cut width, mm	Surface irregu-larities, mm
Manual	5-25	3-4	0.5-1.0
	25-50	4-5	1.0-1.5
	50-100	5-6	1.5-2.0
	100-200	6-8	2.0-3.0
	200-300	8-10	3.0-4.0
Machine (standard to-lerances)	5-25	2.5-3	0.1-0.25
	25-50	3-4	0.25-0.5
	50-100	4-5	0.5-1.0
	100-200	5-6.5	1.0-1.5
	200-300	6.5-8	1.5-2.5

Using a standard ACШ-1 hinge-type cutting machine,
G. N. Klebanov and the authors have carried out a series of expe-
riments on structural steel Grade 40 with a thickness of 6, 12 and
20 mm in the Welding Laboratory of the Bauman Technical
School. As has been found, when the prescribed cutting procedure
is closely followed, plates 6 to 20 mm thick measuring 200 mm
square can be straight-line cut so that the kerf is not more than
0.2 mm out of line longitudinally, the cut surfaces are true as to
cross-sectional squareness to within 2 degrees, the surface irregu-
larities on the kerf face do not exceed 12 to 15 microns in height,
and the cut edges are melted not more than 0.5 mm. As the piece
to be cut grows in size and the length of the cut extends to 500
or 600 mm, the maximum height of surface irregularities on the
kerf faces increases to about 40 microns for straight-line cuts and
65 to 70 microns for shape cuts. Variations in oxygen purity
within 98.5 and 97.5 per cent have not been found to affect the
smoothness of the cut in the thickness range under study.

FLUX-INJECTION CUTTING

1. General Principles

The technique of flux-injection cutting is based on the use of fluxing powders introduced into the kerf to aid in the melting or dissolving and removal of refractory oxides from the cut.

As employed in the Soviet Union, the field of flux-injection cutting may be broken into three distinct methods differing in the action of the flux, namely:

(1) the thermomechanical method;
(2) the chemical method;
(3) the chemo-mechanical method.

Of the three methods, the most commonly used is the thermomechanical method. In this method, the flux is a fine-grained powder of iron or other metal in pure form, such as in the cutting of high-chromium steels, or with additions of some fluxing elements, such as ferrophosphorus in cast-iron cutting. Burned with the oxygen jet, the metal powder adds much heat to that supplied by the preheating flame and the oxidation of the metal, thus making the refractory oxides fluid. At the same time, the metal particles issuing from the tip at a high velocity remove the molten oxides from the cut mechanically, leaving smooth kerf faces. When fluxing elements are added to the metal powder, the thermal and mechanical action is supplemented by the cleansing action of the additions.

The chemical method has not found any appreciable field of application due to lack of safe chemicals. In this method, the flux introduced into the cut does not produce much heat. Instead, it turns the refractory oxides into soluble compounds and helps remove them into slag through chemical action. Chemical fluxes may be salts of sodium, such as sodium carbonate with an addition of 1 to 1.5 per cent of sodium phosphate.

In the chemo-mechanical method, the flux is sand which both dissolves the oxides and removes the molten slag mechanically through its eroding action, thereby producing smooth kerf faces. Sand is cheap as flux, is easy to obtain, produces a very limited

heat-affected zone, and liberates little heat in the reaction zone. The latter factor is especially important in cutting chrome-nickel stainless and heat-resistant steels, as the precipitation of chromium carbides is reduced and the anticorrosion properties of the metal near the cut are practically unaffected by cutting.

On the other hand, sand is very detrimental to the operator's health, as it produces much dust which may give rise to silicosis. The cutting rate in the case of sand is lower than with metal powders or with mixtures of powdered iron and fluxing ingredients. For these reasons, its use is limited.

In all cases, flux may be carried to the cut either by the cutting oxygen jet or by a separate jet of oxygen or air.

The range of metals cut by the flux-injection technique includes those which are hard to cut by ordinary flame cutting. These above all are high-chromium and chrome-nickel heat-resistant and stainless steels, grey iron and ferrous metals and alloys.

In the Soviet Union, the procedures, techniques and equipment for flux-injection cutting have been developed by the Welding Department of the Bauman Technical School, the Autogenous Welding Research Institute and the Krasny Oktyabr Factory. The workers responsible for the development of work have been awarded a State Prize.

2. Equipment for Flux-injection Cutting

Flux-injection cutting equipment uses both single- and double-line cutting oxygen systems. In single-line systems, flux from the hopper is injected into the cutting oxygen orifice by the cutting jet under a pressure varying with the thickness of the piece to be cut. In double-line systems, flux is carried as far as the head by a separate jet of oxygen under a low gauge pressure, where it is injected into the cutting orifice by the cutting oxygen which enters the central passage of the injector built into the torch head (Fig. 191). The pressure of the cutting oxygen is adjusted in accordance with the metal thickness to be cut.

The single-line system is advantageous in that it uses a very simple flux feeder and one handwheel on the hopper with which to admit, shut off or adjust the cutting jet. The torches used with single-line equipment are ordinary cutting torches with minor modifications, such as steel instead of brass in oxygen passages and orifices so as to reduce their attrition by the flux.

On the other hand, double-line systems are more convenient in that the pressure in their flux feeders remains low in the course of normal operation, irrespective of the thickness being cut, because the cutting oxygen is fed under a sufficiently high pressure

directly into the torch head and not into the flux injector. This advantage can, however, be fully utilised only if there is a non-return valve on the torch, flux-supply hose or hopper outlet, as the flux blocking the cutting orifice may instantaneously raise the pressure in the flux-supplying hose.

In addition to injector-type flux feeders, there are flux feeders incorporating cyclone chambers. Their advantage is that they can
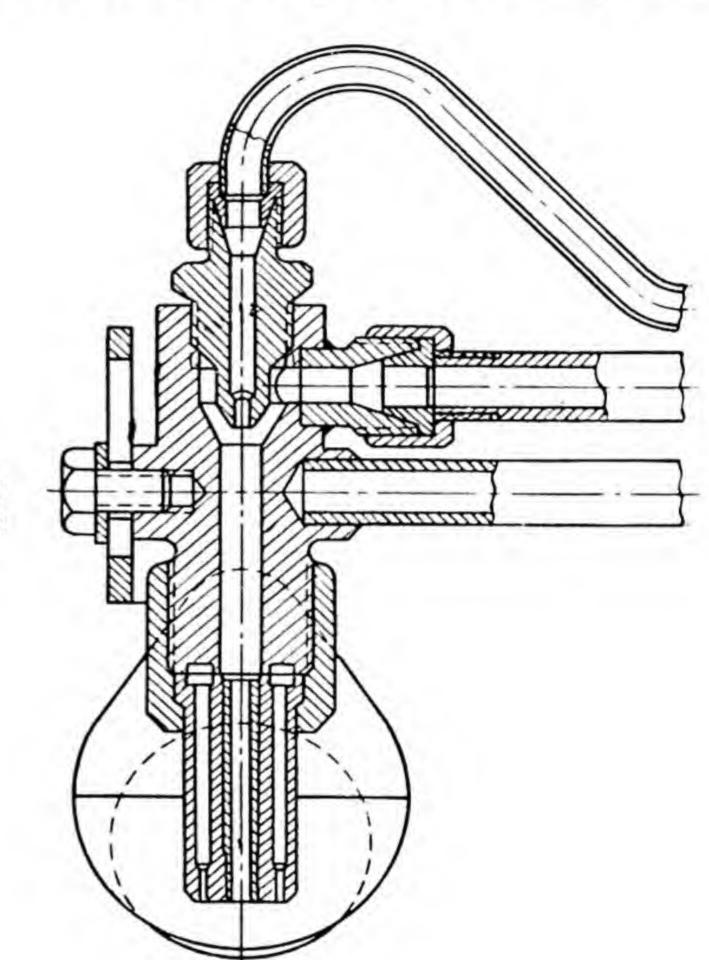
Fig. 191. A cross-section through a torch head with a double oxygen line

use very fine powders which burn better in the reaction zone and thus add to the cutting rate. According to the Autogenous Welding Research Institute, cutting with a cyclone-type flux feeder and a torch to which flux is fed externally is as efficient as the conventional flame cutting of low-carbon steel.

An example of an injector-type single-line equipment is provided by the УФР-2 outfit developed at the Bauman Technical School (Fig. 192). The outfit is intended to flux-cut high-chromium and chrome-nickel steels from 3 to 100 mm thick. Using suitable flux, the outfit may also be employed to cut light-gauge cast iron and brass. The outfit incorporates a flux feeder mounted on a support, and two torches, one for manual and the other for machine cutting.

The flux feeder has a leverage system with which it is possible to admit flux and cutting oxygen to the torch and to adjust their rate of flow.

The flux feeder consists of a hopper *11* to which is attached a tapered chamber *16*. The hopper is made fast to the cover by a hermetically sealed flange. When the outfit is inoperative, the flux outlet is shut off by a bell *2*, and the oxygen flow from a cylinder by a valve *5*. Clockwise rotation of a handwheel *7* screws
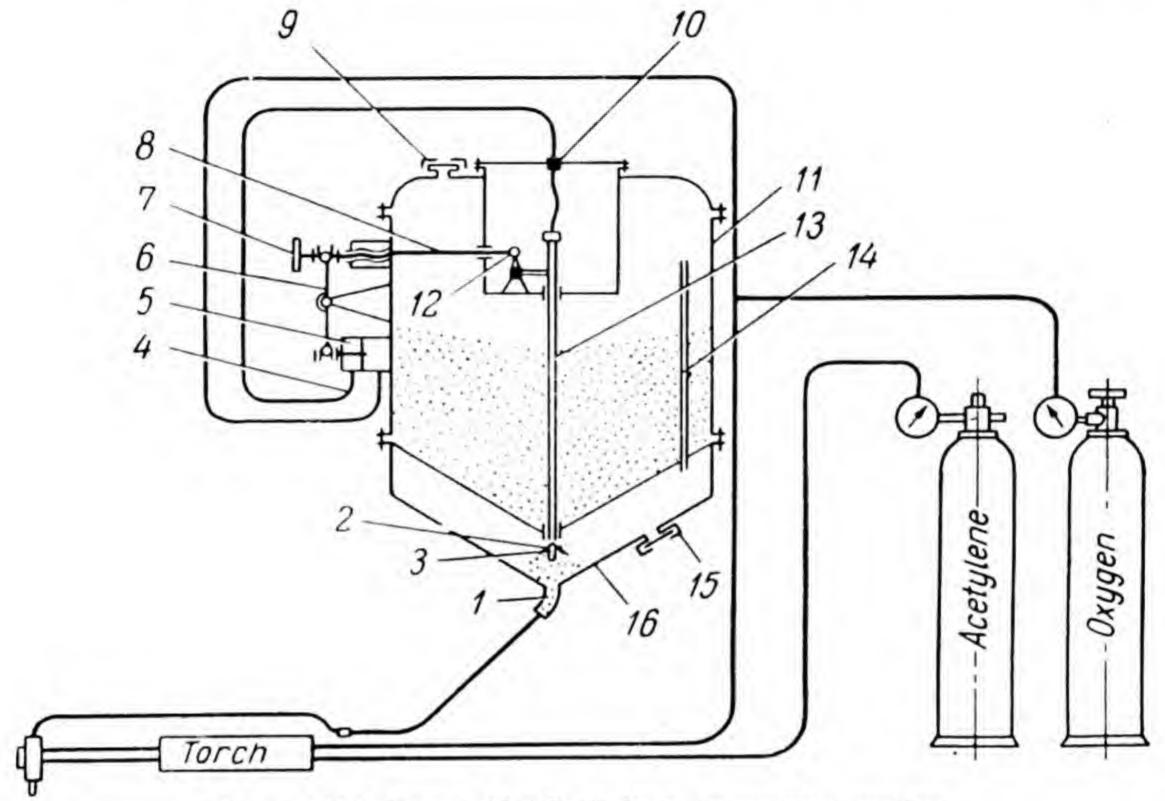
Fig. 192. Type УФР-2 flux-injection cutter

it onto the seal to push a link *8* which transmits motion through a lever *12* to the link *13* of the valve, and the bell *2* goes down to open the flux outlet into the chamber *16*. At the same time, the handwheel *7* bears upon the lever *6* which opens the oxygen valve *5* to admit the cutting oxygen via a connection *4* into the hose which delivers the gas to another connection *10* and the injector nozzle *3* where the high-velocity jet carries along the powder through the connection *1* and into the cutting oxygen orifice in the tip.

When the handwheel *7* is rotated counterclockwise, the valve *5* cuts off the oxygen flow and the bell *2* closes the powder outlet.

The cover of the hopper has a filler neck fitted with a bursting membrane *9* which bursts at a pressure of over 10 atm, thereby protecting the hopper from excessive pressure. Welded into the tapered portion of the hopper is a connecting tube *14* which

equalises the pressure in the hopper and the tapered chamber *16*. Into the sidewall of the tapered chamber is welded an elbow also fitted with a bursting membrane *15* designed for a pressure of 10 atm.

The manual torch of the outfit is a modified Type УР or РР. The torch head is modelled after a machine torch and has a single tip with a central cutting oxygen orifice and six concentrically arranged preheating flames. The torch has an acetylene and a preheating oxygen valve, the flow of the oxygen-flux mixture being controlled by the handwheel on the flux hopper.

The oxygen-flux line of the torch is made from a high-carbon hardened steel to raise its resistance to the abrasive action of the flux. In operation, watch must be kept on the condition of the inserts in the head and tip, or the oxy-acetylene mixture may find its way into the flux hose and hopper.

The torch comes complete with a pair of guide rollers for convenience in traversing the plate.

The machine torch employed in conjunction with the УФР-2 outfit is the double-valve Type РМ-2. To reduce the attrition of the oxygen line by flux, the brass tube supplying the cutting oxygen to the tip is replaced by a steel tube; the inserts pressed into the torch head and tip are also made from high-carbon steel. A general view of the УФР-2 outfit is shown in Fig. 193.

Double-line injector-type systems are examplified by the УРХС-3 outfit (Fig. 194) developed by the Autogenous Welding Research Institute, intended for cutting high-chromium steel up to 400 mm thick, cast iron up to 300 mm thick, brass and bronze up to 150 mm thick, and copper up to 50 mm. It may also be employed for flame machining high-chromium and chrome-nickel steels by the flux-injection technique.

The Type ФП-3 flux hopper *25* is of welded construction and can hold up to 20 kg of flux. Flux is loaded into the hopper through the filler *26* welded to the hopper cover. The tapering bottom of the hopper terminates in a sleeve *19* to which are attached adapters *16* and *18* serving as an intermediate space for the flux falling from the hopper. The pressure in the adapter *18* and the hopper *25* is equalised by means of the tubes *21* and *23* meeting at the joint *22*.

The injector *12* will be found at the bottom of the hopper. It consists of a mixing chamber *14*, an injector nozzle *13* and a seal *17* making the system air-tight.

The quantity of flux admitted into the mixing chamber is adjusted by a handwheel *11* which controls the distance between the injector nozzle and the inlet of the mixing chamber.

The pressure of the flux-bearing oxygen admitted into the injector is adjusted by means of the pressure regulator 7.

The valve 10 serves to equalise the oxygen pressure in the top of the hopper 25 and the injector and to disconnect the two, when necessary.
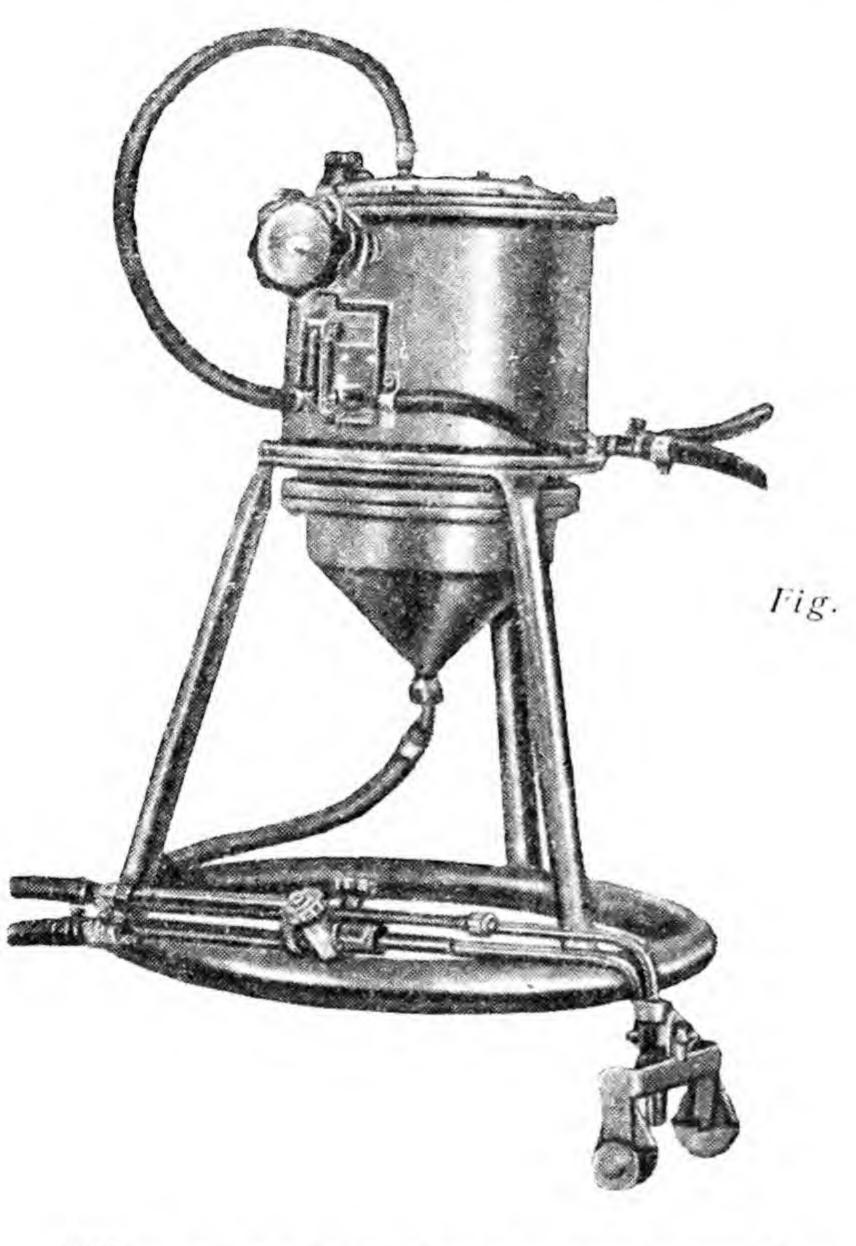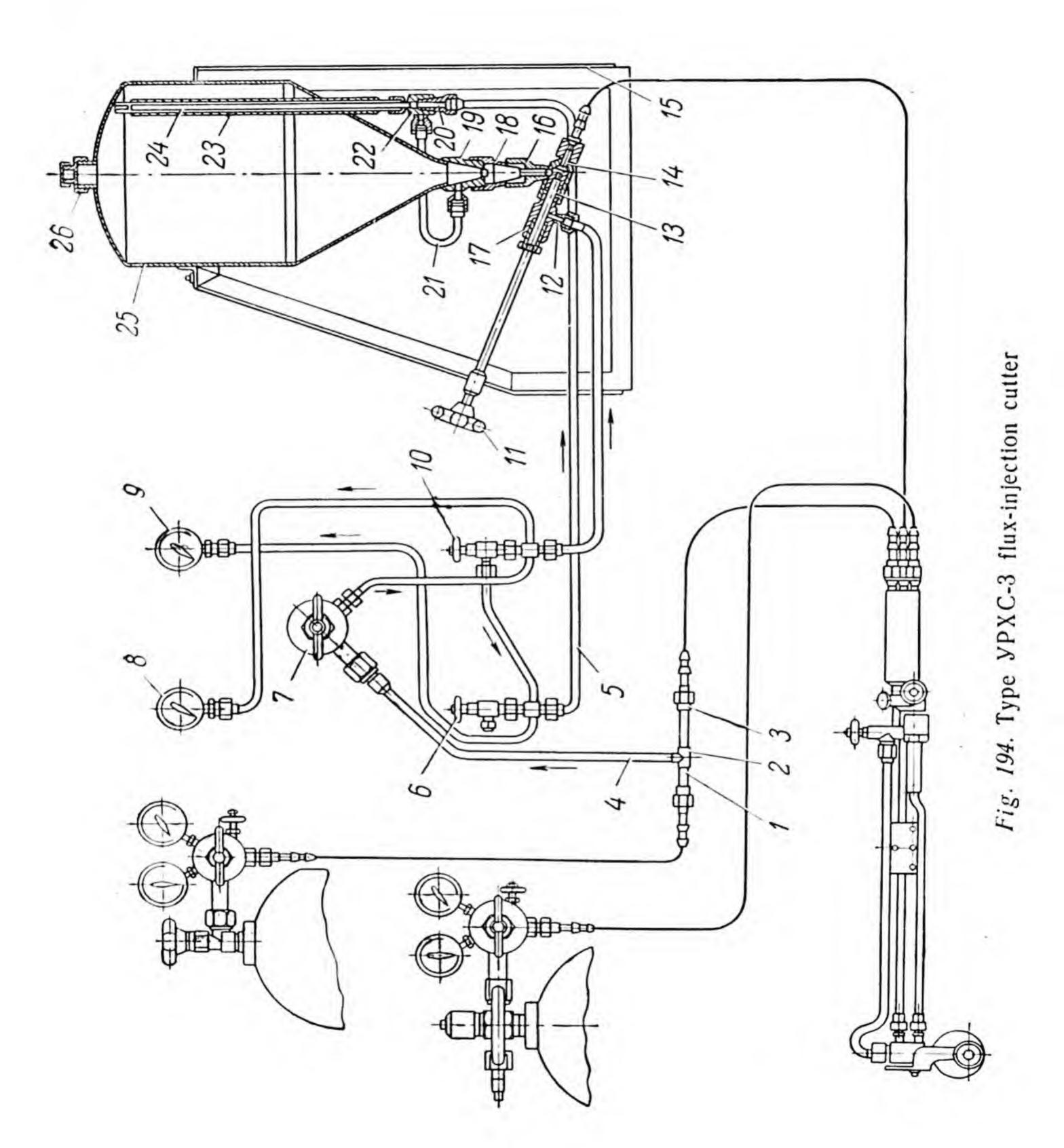
Fig. 193. General view of Type УФР-2 cutter

After operation, the oxygen from the hopper can be released by the valve 6.

When the outfit is operative, oxygen from a cylinder or from a mains enters the flux feeder by a tube *1* and a tee-piece *2* whence it flows in two streams. One stream flows through the connection *3* into the hose and tip to be used as the cutting and preheating oxygen. The other flows up the tube *4* to the pressure regulator 7 where it is brought down to the working pressure and is fed into the injector *12* from which the oxy-flux mixture travels down the hose into the tip.

Fig. 194. Type УРХС-3 flux-injection cutter

Part of the flux-bearing oxygen is fed via the valve *10*, tube *5*, sleeve *20* and tube *24* into the hopper to maintain the flux supply to the tip steady.

All the parts of the flux feeder are mounted on a framework *15*. For convenience, the knobs of the valves and injector as well as the adjusting screw of the pressure regulator are brought out to the front wall of the framework.
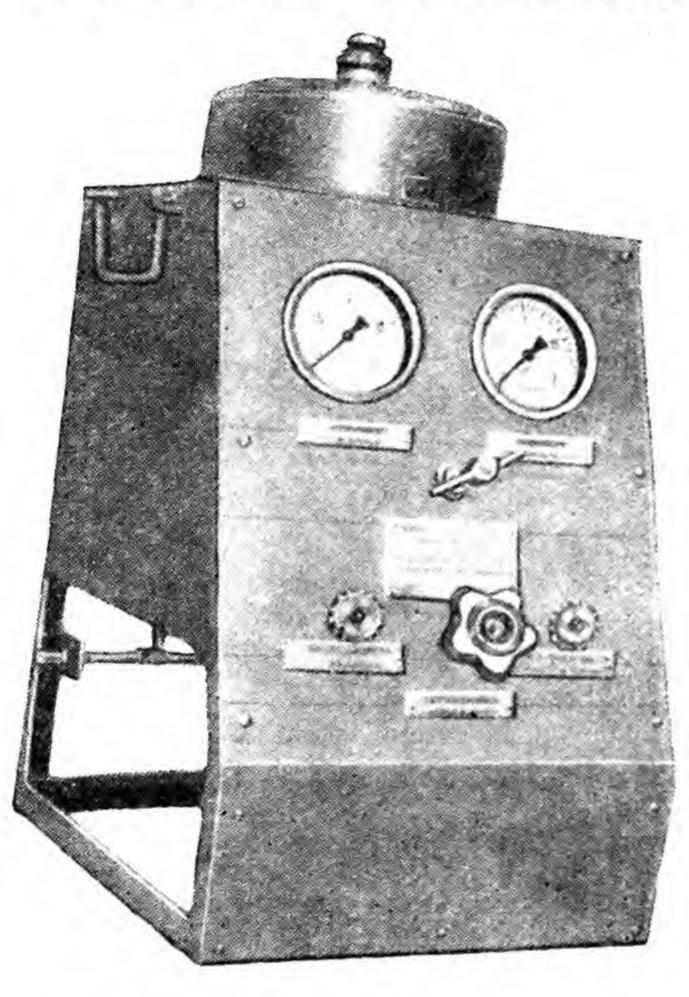
Fig. 195. General view of Type УРХC-3 cutter

The outfit also has two pressure gauges, *8* and *9*, to read the oxygen pressure before the injector and in the hopper. A general view of the УРХC-3 outfit is shown in Fig. 195.

3. Flux-injection Cutting of High-chromium and Chrome-nickel Steels

(1) Difficulties. Steel containing over 5 per cent chromium is very difficult to flame-cut by the conventional technique, if at all.

When the steel is heated and melted, the chromium, because of its high affinity for oxygen, forms Cr_2O_3, a refractory oxide melting at about 2000° C. When the steel is low in chromium, the Cr_2O_3 particles are very few and far between and cannot form

a continuous oxide film which otherwise hampers the progress of cutting and the removal of the molten slag from the kerf. When the chromium in a steel runs high, a continuous film of surface oxide is formed on the heated metal, preventing the underlying layers from being oxidised. Attempts to cut high-chromium and chrome-nickel steels by conventional flame-cutting techniques have usually failed, although use has been made of a powerful preheating flame.

Better, but far from satisfying results have been obtained with a low-carbon steel wire fed into the reaction zone. In such a case, the steel burns as in the normal cutting operation, but the cutting operation lacks in quality and speed.

The arc cutting of high-chromium and chrome-nickel steels with heavily coated low-carbon steel electrodes likewise suffers from the low quality of cuts and low cutting speeds. As the molten steel is not blown out of the kerf, it forms heavy overflow on the bottom side of the cut. Like flame cutting with low-carbon steel wire, arc cutting may only be used for severing, when it is of no importance at all that the resultant cut edges are rough or when they are to be machined afterwards.

The machine-tool cutting of high-chromium and chrome-nickel steels is as labour-consuming and inefficient as its flame or arc cutting. This is especially true of high-alloy chrome-nickel steels which are of high ductility owing to the austenitic structure.

(2) Cutting Procedure. *(a) Preparation.* When cutting sheet steel of high hardness, especially in shape cutting, it should be first heat-treated. The heat treatment varies with the type of steel. Thus, ferritic chromium steel or pearlitic heat-resistant steel (Grades CX8, CX8M, CX12, etc.) should be tempered at 300° C. Austenitic stainless and heat-resistant steels, which display high ductility when hardened, require no preliminary heat treatment, as a rule. They should be annealed only when strain-hardened by cold rolling, as this develops in them both high tensile strength and hardness.

As in ordinary flame cutting, the plate should be flat and cleaned of dirt.

(b) Starting a cut. As in the case of ordinary flame cutting, a small area around the start of the cut is raised to kindling temperature. Practically, the oxy-flux jet is turned as soon as the metal is sweated. In some cases, operators ignore some waste of flux and raise the edge to kindling temperature with the cutting oxygen jet turned on, which is a useful expedient as the flux burns readily.

As a rule, flux cuts should be started at the edge of the piece, as this speeds up the attainment of kindling temperature and produces a straight and smooth starting edge. Where the cut is started away from the edge and inside the profile, the starting hole may be pierced either by the torch (in steel sections up to 30 or 40 mm thick) or by a drill (in the heavier gauges).

(c) The Distance from the Cutting Tip to the Metal. This distance should ordinarily be anywhere from 15 to 20 mm, the actual figure varying with the pressure of the cutting oxygen, i. e., with the thickness to be cut. Experience has shown that when this distance is too short, the cutting operation is sluggish and unsteady as some of the flux is reflected from the metal into the preheating flames to cause popping and even flashbacks. In addition, the tip grows excessively hot, also contributing as a disturbing factor.

(d) The Position and Motion of the Torch. The tip should be directed at an angle of 5 to 10 degrees opposite to the progress of cutting for better contact between the oxy-flux jet and the cut edge and better flux utilisation.

This, however, applies only to straight-line cutting. In shape cutting, the tip should be held at right angles to the surface of the piece.

(e) Speed of Flux-injection Cutting. In the case of high-chromium steels it is somewhat lower than with low-carbon steel cut by the conventional flame-cutting technique.

Flux-injection cutting can probably be speeded up by using fluxes which contain additions which raise the temperature in the reaction zone and facilitate the dissolution of refractory oxides, and by feeding a finely divided iron powder externally to the kerf.

(f) The Preheating Flame. In flux-injection cutting, the heat of the preheating flame has to be divided between the metal, as in conventional flame cutting, and the flux which issues from the cutting orifice with a relatively high velocity and burns in the cutting oxygen. If the heat input from the preheating flame is low, there will be a slow rise to kindling temperature at the start of the cut, and not all of the metal powder which makes up the bulk of the flux will ignite. Instead, it will ignite at a greater distance than is good for the normal progress of cutting.

In practice, the heat input from the preheating flame, as expressed in terms of fuel-gas flow, should be 15 to 25 per cent greater in flux-injection cutting than in conventional flame cutting. The flame should be normal or with a slight excess of acetylene.

(g) Cutting Oxygen Pressure. Irrespective of the equipment used, the pressure of the cutting oxygen before the tip must be roughly equal to that used in the conventional flame cutting of low-carbon steel.

(h) Fluxes. In the Soviet Union, the most commonly used fluxes for cutting stainless steels are the BC and the BK, both straight iron powders manufactured, respectively, by the Tula and the Sulin iron and steel works. Injector-type equipment does not usually use fine-grained powders (such as the BM), as they may ignite while within the tip or hose. This precaution also applies to straight metallic powders containing over 96 per cent iron.

In some cases, iron powders may contain additions of fluxing ingredients, such as mill scale or sand. In the latter case, however, health hazard is very grave, indeed.

Mill scale, when added to an iron powder, may run as high as 25 to 50 per cent. Being a waste of rolling, mill scale is as cheap and easy to get as sand, but is less detrimental to health. Mill scale displays good fluxing properties and produces a relatively narrow heat-affected zone.

Suggested cutting conditions for stainless and heat-resistant steels with injector-type equipment are summarised in Table 58.

Table 58

Suggested Cutting Conditions

Thickness of steel, mm	Tip size for PM-2 torch	Oxygen pressure regulator setting, atm (gauge)	Cutting speed, mm/min	Flux consumption per linear metre, kg	Oxygen consumption per linear metre, cu m	Acetylene consumption per linear metre, litres
3	3	3-3.5	325-375	0.4-0.5	0.4	25-30
5	3	3-3.5	300-325	0.5-0.6	0.5	30-40
10	3	3.5-4	250-300	0.6-0.7	0.75	40-50
20	3	4-4.5	225-275	0.7-.08	0.9	50-60
30	4	4.5-5	200-250	0.8-0.9	1.1	60-70
40	4	5-5.5	175-225	0.9-1.0	1.3	70-80
50	4	5.5-6	150-200	1.0-1.1	1.5	80-90
100	4	7-8	90-130	2.0-2.5	2.0	120-150
150	—	10-12	60-100	3.7-6.6	2.5-4.5	150-200
200	—	12-14	50-75	6-10.3	5-8.5	250-350
250	—	14-15	40-50	8.7-14.7	8-13.5	400-500
300	—	15-16	35-40	12-19	12-20	700-800
400	—	17-18	30-35	23-24	—	—
500	—	19-20	25-30	27-28	—	—

Fig. 196 shows a photograph of parts cut from stainless steel by the flux-injection technique.

Fig. 196. Parts machine-cut by the flux-injection process from stainless steel

(3) Flux-injection Stack Cutting. Stack cutting is the most logical and advantageous application for the flux-injection technique, especially where stacks are 50 mm or more thick. This is because (a) the hottest portion of the flame in this case is very long and can only be fully utilised in the thicker material; (b) in light gauges, a good proportion of the flux is wasted because the iron powder in it burns past the cut; and (c) as a result of these two factors, the efficiency of the cutting operation is higher and the specific flux consumption is lower on the thicker material.

The flux-injection technique makes it possible to cut stacks not clamped by cramps, with the individual plates separated by as much as 6 or 7 mm as the hot flux particles in the cutting jet burn within up to a few centimetres of the cutting orifice and much slag is formed in cutting.

A group of investigators at the Bauman Technical School have developed procedures for stack-cutting stainless steel with the individual plates as thin as 2 or 3 mm.

(4) The Effect of Flux-injection Cutting on the Structure and Anticorrosion properties of Stainless Steel. The studies conducted by the Welding Laboratory of the Bauman Technical School on stainless sheet steel 1X18H9T seem to indicate that flux-injection cutting has but a negligible effect on the structure of the metal. Most of the specimens showed a coarse-grained poly-

herdic structure in the kerf faces and the carbide phase in the heat-affected zone. The metal beyond the heat-affected zone had a dotted structure with some amount of segregated carbides.

The precipitation of chromium carbides in the metal near the kerf due to the cutting heat was limited and did not usually bring about intercrystalline corrosion in the cut edges. Indeed, the cases of intercrystalline corrosion were very few and far between, occurring immediately in the kerf face to a depth of not more than 0.3 mm.

Oil-refinery equipment, vessels, piping and the like, manufactured from stainless steel, come in contact with the aggressive medium at flame-cut edges very seldom. In most cases this contact is along welded joints, usually made by the arc process over the edges prepared by the flux-injection technique. For this reason, the metal area which has been subjected to flame cutting and arc welding in that order is of special interest from the viewpoint of the anticorrosion properties of stainless steel. The large number of specimens examined for this purpose have failed to reveal any traces of intercrystalline corrosion in that metal area. The explanation lies in the fact that the cut edges are remelted during welding to a depth of 1 or 1.5 mm, and the heat-affected zone left over from cutting is overlapped by that of welding.

4. Flux-injection Cutting of Cast Iron and Nonferrous Metals

(1) **Cast Iron Cutting.** The conventional flame technique fails to cut cast iron as the latter ignites in oxygen at a point above its melting temperature. Besides, the kerf faces are immediately coated by the film of the refractory silicon oxide, while the large amount of carbon dioxide evolved in the cutting of cast iron due to its high carbon content contaminates the cutting oxygen, thus reducing its oxidising property.

Attempts have been made to cut cast iron using a preheating flame of increased heat output, or a low-carbon steel rod fed into kerf, or by the arc process with a carbon or a steel electrode. In all cases, however, the cut has been of poor quality and the cutting rate low.

The oxy-flux cutting of cast iron differs little from that of high-chromium steels. The difference boils down to a special flux containing about 30 or 35 per cent blast-furnace ferro-phosphorus. The cutting speed is by 50 or 55 per cent lower than in the case of high-chromium steels. For cast iron sections up to 50 mm thick, the gas and flux consumption is 2.5 to 3 times and for the heavier 1.5 to 2.5 times greater than for high-chromium

steels. The distance between the tip and the metal surface should be kept within 30 to 50 mm. The cutting operation may have a chilling effect on the kerf and the heat-affected zone; nor is

Fig. 197. A cut through a cast-iron round 360 mm in diameter

cracking due to local heating and cooling ruled out. Therefore, when flux-injection cutting is employed to remove gates and risers from iron castings or to cut them into blanks for further

Table 59

Flux-injection Cutting of Cast Iron

Thickness of cast iron, mm	20	50	100	150	200	250	300
Approximate cutting rate, mm/min	80-130	60-90	40-50	25-35	20-30	17-25	15-22
Oxygen consumption, cu m per linear metre of cut	0.9-1.8	2-4	4.5-8	8.5-14.5	13.5-22.5	20-31.5	27.5-43
Acetylene consumption, cu m per linear metre of cut	0.1-0.16	0.16-0.25	0.3-0.45	0.45-0.65	0.6-0.87	0.75-1.1	0.9-1.3
Flux consumption, kg per linear metre of cut	2-3.5	3.5-6.0	6-10	9-14.5	11.5-19.0	14-23	17-28

fabrication (as distinct from the severing of cast iron scrap for remelting), the cast iron piece should be given a preheat and allowed to cool slowly. It is good practice to remove gates and risers right in the foundry, while iron castings are still sufficiently hot.

Table 59 compares the conditions of flux-injection cutting for various thicknesses of cast iron. Fig. 197 shows a cast iron round 360 mm in diameter severed by the flux-injection technique as the piece was rotated at a uniform speed.

(2) Copper Cutting. Before the advent of the flux-injection technique copper and its alloys had been considered uncuttable by flame at all. Instead, machine-tool cutting and shearing had to be relied upon in the case of light gauges, or drilling and subsequent chieselling in the case of the heavier sections.

Among the factors that make the cutting of copper and its alloys impossible are the low heat of oxidation, high thermal conductivity (especially of copper) and the high melting point of oxides.

In its basic features, the flux-injection cutting of copper and its alloys is similar to that of high-chromium steels, the differences being in the flux and cutting conditions employed. Fluxes for copper, brass and bronze are mixtures of powdered iron, powdered aluminium (15-20 per cent) and ferro-phosphorus (10-15 per cent).

The copper-alloy piece to be cut is given a preheat of 200 to 400° C. The tip-to-metal distance varies with the metal thickness to be cut anywhere from 30 to 50 mm. Cuts inside a profile are started by drilling a hole with a drill 10 to 20 mm in diameter, depending on the material on hand.

Table 60

Flux-Injection Cutting Table for Copper

Thickness of copper, mm	10	20	30	50
Approximate cutting rate, mm/min	110-180	65-80	35-50	20-25
Oxygen consumption, cu m per linear metre of cut	1.0-1.3	1.7-2.4	3-4.5	6.5-10
Acetylene consumption, cu m per linear metre of cut	0.12-0.15	0.22-0.28	0.4-0.5	0.83-1.1
Flux consumption, kg per linear metre of cut	1.7-3.2	3.4-6.2	6.5-12.7	17-30

Tables 60 and 61 compare the cutting conditions for various thicknesses of copper and copper alloys.

Table 61

Flux-injection Cutting Table for Brass and Bronze

Metal thickness, mm . .	10	20	50	80	100	150
Approximate cutting rate, mm/min	225-140	190-120	120-80	75-45	55-30	25-15
Oxygen consumption, cu m per linear metre of cut	0.5-1.0	0.6-1.2	1.2-2.4	2.2-5.0	3.6-8.5	10.5-21.5
Acetylene consumption, cu m per linear metre of cut	0.08-0.12	0.09-0.16	0.15-0.23	0.25-0.4	0.35-0.6	0.8-1.3
Flux consumption, kg per linear metre of cut	1.7-3	2-3.5	3.5-6	6-11	9-17	21-33

Notes: 1. The table data are valid for an oxygen purity of not below 99 per cent.
2. The higher cutting rates and lower gas and flux consumption apply to machine cutting; the lower rates and higher consumptions to manual cutting.

In Fig. 198 is shown the cut edge of a brass plate 85 mm thick.

Fig. 198. A cut through a brass plate 85 mm thick

Heavy sections of cast iron and copper alloys may be cut with УРХС-3 and УФР-4 outfits, while the lighter gauges (cast iron under 50 mm and copper alloys under 20 or 25 mm) with the УФР-2 outfit.

CHAPTER XXV

UNDERWATER OXY-CUTTING

Though it has a relatively modest field of application, underwater oxy-cutting has acquired considerable value as an efficient operation in ship repair, ship salvage, rescue work and construction. Two varieties of underwater oxy-cutting have been especially widely employed in the Soviet Union to date. These are oxy-gas cutting and oxy-arc cutting. In the former case, the metal to be cut is heated to kindling temperature by a preheating flame as in ordinary oxy-gas cutting. In the latter case, the arc drawn between the work and a metallic or a carbon electrode does the job.

The preheating flame and the cutting oxygen must be shielded from contact with water. This is usually done by surrounding the tip with a mantle or a bell into which is led a compressed jet of air, or of nitrogen or even of oxygen, and the partially or fully oxidised gases (CO and CO_2) of the preheating flame.

Because of the cooling effect of water, the heat input from the preheating flame must be 10 to 15 times greater than in "on land" cutting. Likewise, the oxygen and fuel-gas pressure at the tip outlets must be higher to overcome the water pressure.

The fuels used for underwater cutting are mostly hydrogen and petrol. Acetylene has been used on a limited scale because of its tendency to explode under increased pressures.

Fig. 199 shows a Type УПР oxy-hydrogen unit intended for underwater cutting. It consists of a special-purpose torch *1*; an electric lighter *2* to start the preheating flame under the water; hoses *3* to supply hydrogen, oxygen and air to the torch; a control panel *4* housing the pressure regulators and the lighter control; and three manifolds *5* (one each for hydrogen, oxygen and compressed air). The shield around the preheating flame and cutting jet uses compressed air or oxygen fed under the outer bell.

The УПР unit has been designed for steel sections up to 70 mm thick and for operation at shallow depths (not over 3 m). The working pressures of hydrogen and air are set in accordance

with the depth and do not exceed 3.5 atm (gauge) for hydrogen and 5 atm (gauge) for air.

The pressure of the cutting oxygen varies with the metal thickness on hand, but does not exceed 10 atm (gauge).

The increased fuel-gas consumption in underwater cutting and the low capacity of cylinders handicap the use of hydrogen for this purpose. This likewise applies to gaseous hydrocarbons
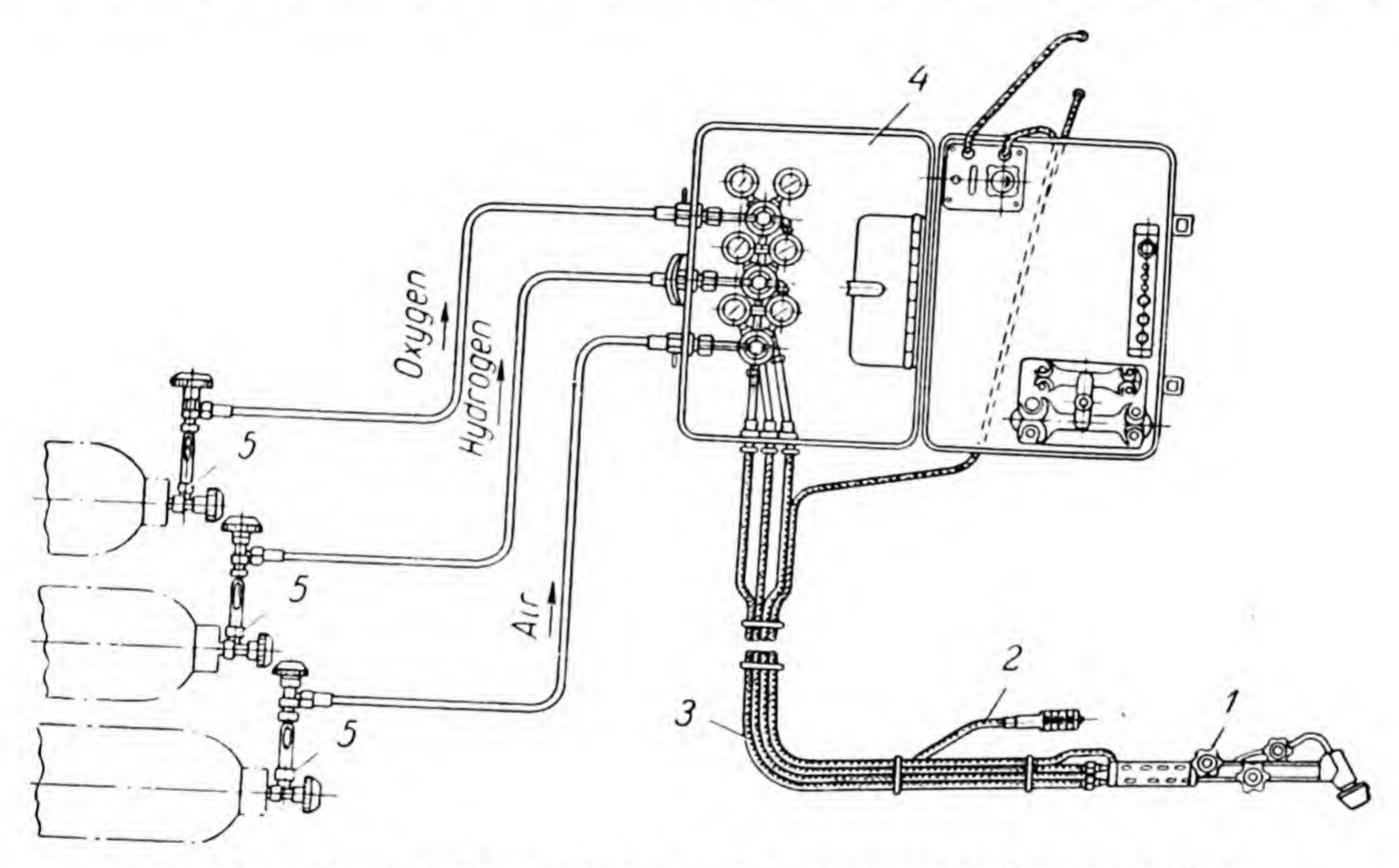
Fig. 199. Type УПР outfit for underwater oxy-hydrogen cutting

(such as methane and natural gas), despite the fact that their calorific value is greater than that of hydrogen. This is why petrol has proved to be more advantageous and economical where large-scale underwater cutting is involved.

In the latest makes of oxy-benz cutters, petrol is fed under a considerable pressure via spiral passages of small bore into a specially designed mixing chamber where the fuel is atomised by the preheating flame oxygen jets, is vaporised and ignited. The petrol vapours completely burn in the outer envelope of the preheating flame. The pressure in the service fuel tank from which the petrol is fed to the torch is maintained by nitrogen supplied via a pressure regulator from a cylinder.

One commercial unit for underwater cutting with the oxy-benz flame, designated the БУПР (developed by the Autogenous Welding Research Institute), consists of an oxy-benz torch, a

control panel, an eight-cylinder oxygen manifold, armoured oxygen and petrol hoses, and an electric lighter operating from a storage battery. The outfit will cut steel sections up to 100 mm at a depth of down to 30 m. The gas pressure is adjusted to suit the metal thickness on hand and the depth assigned. The maximum pressure for oxygen is 15 atm (gauge) and for air 12 atm (gauge).

The oxygen consumption in oxy-benz underwater cutting is 30 to 60 m^3/hr, and that of petrol 10 to 20 kg/hr in continuous operation. The actual consumption of both oxygen and petrol is much greater due to the greater (though inevitable) waste of time by the diver while the preheating flame keeps burning. The nitrogen consumption in underwater cutting is low, as it only goes to build up pressure inside the service fuel tank.

An advantage of oxy-benz underwater cutting is that the flame provides a greater heat input and much CO and CO_2 is formed for use in the bell, so that no additional amounts of air or oxygen are necessary.

FLAME MACHINING

1. Basic Features

In flame machining, the cutting tip makes an angle of not more than 35° with the surface of the work, while in ordinary flame cutting the angle may be anywhere from 45 to 90°. Instead of severing the metal, the oxygen jet issuing from the machining tip removes some metal from the surface of the piece. Diagrammatically, flame machining is shown in Fig. 200. As distinct
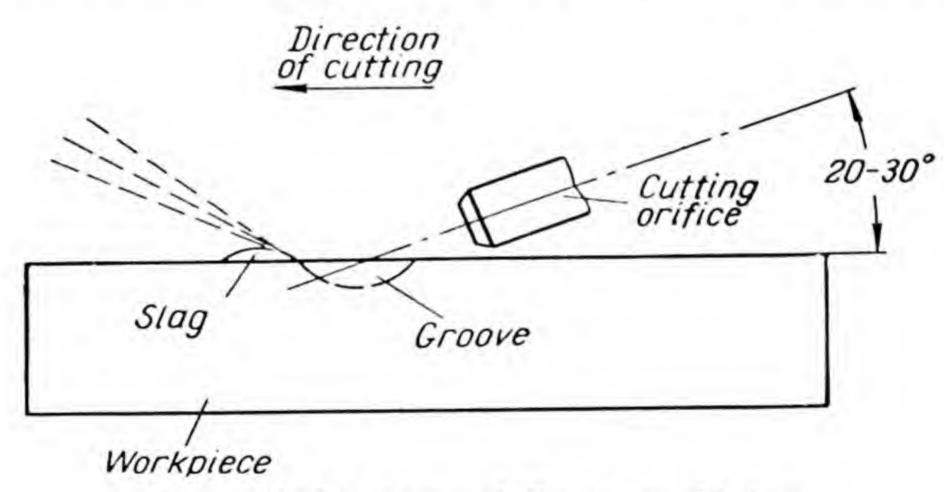
Fig. 200. Diagram of flame machining

from ordinary cutting, the slag formed in flame machining contains much more unburnt iron.

The tilt of the tip (or the flame angle), the low pressure (3 or 4 atm, gauge) of the cutting oxygen and the low velocity of the cutting jet serve to deform the flame considerably. As the tip is advanced at a much greater rate along the line of cut, very little stock is removed. The best results are obtained when the tip travels in the direction of the oxygen jet and the resultant oxides are forced ahead in the line of cut by the cutting jet. Should the tip be moved at right angles to the oxygen jet, the flame-machining operation would stop immediately, as the molten slag would be forced out of the cut sideways, without preheating the metal ahead of the tip.

2. Types and Applications of Flame Machining

Flame machining is widely employed in all metal-making and metal-working industries to perform planing, milling, turning, drilling and boring operations.

Planing includes surface planing, deseaming and hogging. In surface planing, the tip operates like a standard cutting tool and removes the desired stock either in a single pass or in several passes. In the former case, use may be made of several tips clamped on the saddle of a machine. In addition to flat cuts, oval cuts may be made, usually by a single pass, the shape and size of the cut varying with the cutting conditions and flame angle. *Deseaming* is widely used for removing cracks or seams in blooms, billets, slabs and rounds on their way to the various finishing departments of steel mills, or in the steel foundry for exploring and cleaning out cracks and sand holes. Hogging is used to remove superfluous metal in steel castings and also risers and sprues where they are difficult or impossible to remove by ordinary flame cutting. Both deseaming and hogging may be performed either manually or by automatic machines. One such machine is the CKC-54 developed by the Autogenous Welding Research Institute. The production rate of this machine (based on machine time only) is 8 to 12 m^2/hr, when run continuously.

Milling covers those cuts which resemble the ones produced by rotary mills in tool machining. Flame milling is mainly employed for edge or joint preparation prior to welding, for removing defective welds, and for back-milling prior to depositing the sealing run on the reverse side of a weldment. When used for edge preparation in combination with flame cutting, flame milling produces a shallow oval cut on the surface, and the rest of the metal is removed from the edges by a cutting tip. This technique offers the advantages of greater production rates, better cut quality and appreciable oxygen savings.

Turning includes cuts similar to those produced on a standard tool lathe, with the work rotating about its axis. The tip may be either advanced along the work (screw feed) to remove the desired stock along the piece, or left stationary (zero feed) to produce straight-sided cuts all the way round the work. Sometimes, flame turning may be employed for roughing rounds or for cutting flat threads.

Drilling is used for making blind or through holes in many applications (for centring, subsequent billet piercing, rivet cutting, removal of fronzen bolts, etc.). The tip is held at right

angles to the surface of the work and is moved either forward or to and fro in the direction of the oxygen jet.

3. Torches for Flame Machining

In construction flame-machining torches differ from cutting torches in that:

(a) they have larger passages for the preheating flame, as flame machining requires more intensive preheating;

(b) they have larger oxygen passages and, notably, a larger cutting oxygen orifice;

(c) they are longer, sometimes up to one metre long.

Several types of flame-machining torches are available on the market in the Soviet Union. Some of them are intended for use in metal-working industries and others in metal-making industries. The former usually flame-machine steel in a cold state, without the plate or piece being preheated, as in edge preparation for welding, removal of defective welds or local flaws, etc. The latter flame-machine the metal in a hot state, such as in deseaming, hogging, etc.

All torches are, in addition, classed into manual and machine. The latter are mainly used where long cuts are involved, such as in edge preparation for welding, deseaming, etc.

In some cases torches may have a bracket and a director nozzle to feed low-carbon steel wire into the reaction zone so as to cut down the time necessary for bringing the metal to kindling temperature at the start of a cut. The wire, which is usually up to 500 mm long and 6 mm in diameter, is fed manually. The end of the wire is introduced into the flame and melted on the surface of the metal. Then the cutting oxygen jet is turned on by a lever valve, and the white-hot tip of the wire ignites. In burning, it liberates enough heat for the cut to be started. The feed of the wire is stopped as soon as the cut progresses all by itself.

The use of low-carbon steel wire is especially advantageous when, instead of acetylene, coke-oven or any other substitute gas is burned in the preheating flame. The reason is that with coke-oven gas it usually takes 20 to 30 per cent more time to raise the start of a cut to kindling temperature. The additional heat of the wire cuts down this time and the consumption of the fuel gas. The coefficients for the substitution of acetylene by other gases are given in Sec. 4, Chapter VIII.

Ordinary oxygen pressure regulators are useless in flame cutting as they fail to provide for the necessary adjustment of the preheating flame. This is because the rate of oxygen flow is very high and the loss of gas pressure at the outlet from the low-

pressure chamber of the regulator is considerable. Instead use should be made of special regulators with increased passages, such as the PK-50.

The performance data on flame-machining torches are summarised in Table 62.

Table 62

Performance Data on Flame-machining Torches

Tip size	Cut width, mm	Cut depth, mm	Oxygen pressure before tip, atm (gauge)	Machining rate, m/min	Appr. weight of removed stock, kg/min	Gas consumption					
						m³/hr			lit/kg of removed metal		
						oxygen	acetylene	coke-oven-gas	oxygen	acetylene	coke-oven gas
1	15-30	2-12	3-6	1.5-8	1.0-3.0	18-40	0.9-1.0	4-5	180-300	5-17	20-80
2	18-35	2-16	3-8	1.5-10	1.2-3.5	20-55	0.9-1.0	4-5	160-250	4-14	19-70
3	30-50	2-20	3.5-10	1.5-10	2.6-4.5	30-75	0.9-1.0	4-5	220-300	3-7	19-32

Note: The table data apply to an oxygen purity of 98-98.5 per cent and oxygen hoses 9 to 10 m long and 9.5 mm in inside diameter.

4. Flame-machining Technique

As in ordinary flame cutting, the start of a machine cut has to be raised to kindling temperature. As has already been noted, this can be facilitated by feeding a length of low-carbon steel wire into the reaction zone.

Another starting technique uses a nick or a burr punched at the start of a cut. Naturally, the chip and burr can be brought to ignition quicker than the solid metal.

Until the metal ignites, the tip should be held at 70 to 80° to the surface. As soon as the metal is ready to burn, the tip is directed to make an angle of 10 to 30° with the piece, the cutting oxygen jet is turned on, and the torch is advanced at the desired speed and in the desired direction.

For the flame-machining operation to progress normally, it is essential that a proper ratio be maintained between the cutting speed and the rate of oxygen flow (see Table 62). When the machining rate is too high and the rate of oxygen flow too low, the amount of slag produced proves insufficient for preheating the metal ahead of the cut, and the progress of cutting is interrupted. When the machining rate is too low and the rate of oxygen flow too high, the surface of the cut is rough, while too much slag produced may drown the tip and put eut the flame.

The flame angle varies with the depth of the cut required, being greater for deeper cuts.

If the metal is to be heated most efficiently and the cutting oxygen jet is to oxidise the metal in the best possible way, the tip must be held so that the inner cones of the preheating flames
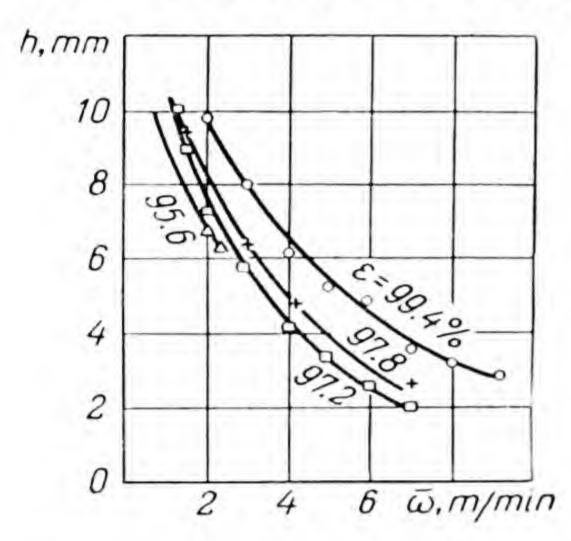

Fig. 201. Depth of machining, h, as a function of cutting speed, $\overline{\omega}$, and various oxygen purities, ε. Cutting conditions: rate of cutting oxygen flow $V_{ox} = 20.2$ m³/hr; flame angle $\alpha = 20°$; diameter of the cutting oxygen orifice: $d_{out} = 5$ mm

closest to the metal nearly touch the latter. The least permissible distance between the tip and the metal depends on the construction and outside diameter of the tip. In some makes, the tip has a steel ring put on, which touches the metal being machined.

Oxygen purity is a factor which affects cut size, machining rate and specific oxygen consumption in a very obvious way. An

Fig. 202. Appearance of a flame-machine surface

increase of one per cent in oxygen purity (between 97 and 99.5 per cent) increases machining rate and reduces specific oxygen consumption by 15 to 20 per cent. When oxygen purity falls to below 95 per cent, the flame machining of low-carbon steel becomes practically unfeasible. Fig. 201 relates the depth of cuts to machining rates at various oxygen purities.

In flame machining, the preheating flame is important mostly at the beginning of a cut. During the machining operation, the preheating flame may in most cases be turned off, as the molten slag supplies the necessary heat instead.

5. Flux-injection Flame Machining of Stainless and Heat-resistant Steels

Flux-injection flame machining is remarkable for high production rates. It removes three times as much metal as the oxy-arc process and eighteen times the metal removed by an air-operated chisel.

Flux-injection flame machining is mostly limited to the deseaming (removal of cracks, tears, sand holes and gas pockets) of ingots from stainless or heat-resisting steels prior to rolling. Where defects are found only at some places on the surface of an ingot, this may be done manually. Where defects are revealed over a considerable area, the operation is usually mechanised.

As in the case of the ordinary flame machining of low-carbon and low-alloy steels, the flux-injection machining of stainless and heat-resistant steels involves high speeds of torch travel and large oxygen consumptions. The torch usually makes an angle of 20 to 45° with the surface of the work.

Flux consumption in flux-injection machining is smaller than in flux-injection cutting, as the machining jet is better utilised. This same factor is responsible for higher production rates in flux-injection machining.

The flux-injection machining technique may use either manual machining torches (Table 63) or specially designed machines, like the УКФ-3 developed by the Autogenous Welding Research Institute.

Table 63

Suggested Conditions for Manual Flux-injection Flame Machining of Stainless Steel

Flame angle, deg.	Tip-to-metal distance, mm	Cut width, mm	Cut depth, mm	Specific consumption	
				oxygen, cu m per kg of removed metal	flux, kg per kg of removed metal
20	8-10	20-24	2-3.5	0.65-1.0	1.5-2.0
25	14-18	20-24	3-4.5	0.55-0.8	1.0-1.6
30	18-28	22-24	4-4.5	0.55-0.65	0.8-1.3
45	32-45	24-26	3-4	0.65-0.7	0.7-1.1

Note: The table data apply to flux grains passing Nos. 0105 and 025 sieves, a flux-supply hose 5 or 6 m long and an oxygen purity of 98.5-99 per cent.

OXYGEN LANCE CUTTING

Oxygen lance cutting is a process by which holes are pierced in heavy blocks of metal by a jet of oxygen passing through a steel pipe whose end is raised to kindling temperature and is placed against the surface of the piece to be cut (Fig. 203).
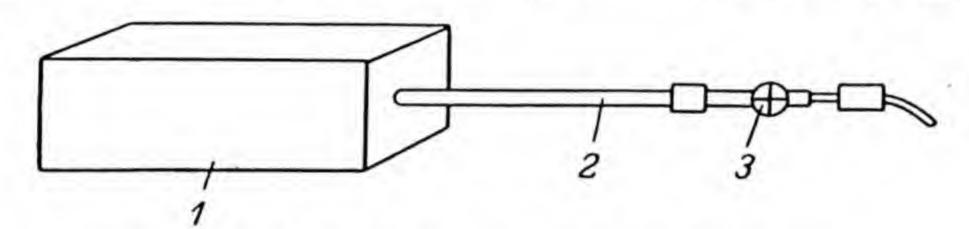
Fig. 203. Diagram of an oxy-cutting lance:
1 — metal to be cut; *2* — lance; *3* — handle and valve

A lance cut is started by raising the tip of the pipe, called the lance, and the point of cutting or combustion to kindling temperature. This may be done with a welding torch, a carbon arc or by using the heat generated when current is passed from a welding source through a carbon plate clamped between the lance and the work.

After the lance is raised to the required temperature, oxygen is made to pass through the lance under a gauge pressure of 1 or 2 atm. This causes the lance tip and the heated spot on the work to ignite. The oxygen pressure is raised to 5 or 6 atm, and the lance is advanced into the workpiece. From this moment on, the cutting operation is sustained by the heat liberated by the oxidation of the metal. The slag formed during the operation flows out of the hole through the clearance between the walls of the hole and the lance. The lance shoud be moved to and fro from time to time, or the hole may be filled with slag. For ease of slag removal, the work should be held in an inclined position or vertically and the cut should be run uphill. As often as not,

however, the work has to be held horizontally despite the difficulty with slag run-off.

As the lance advances, it should be given a half-turn in each direction periodically, or else the hole will be distorted in outline.

The lance may be made from low-carbon steel. The cross-section of the lance varies with the depth of the hole to be pierced as well as with the pipe sizes available. According to the Welding Laboratory of the Bauman Technical School, good results have been obtained with a thick-walled lance with an outside diameter of 17 to 19 mm and an inside diameter of 8 to 6 mm, respectively, with a low-carbon steel rod 5 mm in diameter inserted in the pipe (Table 64). Far worse results have been obtained with a lance made from a thin-walled gas pipe 12 mm in diameter with three 6-mm low-carbon steel rods inserted in the lance. The thicker pipe and rod inserts both supply more fuel and heat, while the rods reduce the area of the effective opening in the lance, thereby imparting greater velocity to the oxygen jet while reducing its consumption.

The effect of oxygen lance cutting on the structure of steel varies with the latter's analysis and susceptibility to hardening Investigations have shown that steels with an increased content of alloying elements, such as chrome-molybdenum steels, are susceptible to cracking in the area near to lance-cut holes owing to their tendency to harden. This is why, prior to lance cutting, such steels should be given a preheat of at least 300° C. No cracks develop in steels containing 0.3 or 0.4 per cent carbon, and they may be lance-cut without preheating.

The applications for oxygen lance cutting include the severing of plugs in blast-furnaces, the removal of risers and sprues from steel castings, the making of centring holes in shafting, the opening of tap holes in blast-furnaces, the burning of holes in blocks of metal to be broken up by blasting, the piercing of starting holes for heavy cuts inside the periphery of a profile, etc.

The principal drawback of oxygen lance cutting is that every centimetre of hole burned by a lance requires 2.5 to 5 centimetres of thick-walled piping or twelve centimetres of thin-walled piping (Table 64).

Of late, the trend has been towards using nonconsumable lances and metallic powders (iron or iron + aluminium) fed into the reaction zone through the pipe. The metallic powders supply much more heat than a burning lance; in addition, there is no need for replacing a burnt lance, either, which otherwise interrupts the progress of cutting. Such powder-using lances are especially advantageous in burning holes in stainless steels.

Table 64

Suggested Conditions for Oxygen Lance Cutting
(after the Bauman School)

Type of lance	Direction of cutting	Dia. hole, mm	Average of time, min., mm per linear metre ot cut	Average oxygen consump- tion, m³/m	Average lance con- sumption, m/m
Seamless steel pipe, 17 mm OD/8 mm ID; steel wire inside 5 mm dia.	Horizontal	40-50	8.9	7.9	5.1
½-inch gas pipe; three steel rods inside. 5 mm dia. each	Ditto	60-70	21.3	10.0	12.7
Seamless steel pipe, 19 mm OD/6 mm ID, with steel wire inside 5 mm dia.	Uphill at 10° angle from the horizont- al	40-50	—	5.4	4.6
Ditto	Vertical	20-25	3	2.8	2.7

Calculations of standard time and standard output in oxy-gas welding. Accident prevention

CHAPTER XXVIII

STANDARD TIME AND STANDARD OUTPUT IN OXY-GAS WELDING

Under the system adopted in the Soviet Union, the performance of a worker is assessed in terms of standard time and standard output.

The standard time is the time in which an average worker is able to make a piece of work. The standard output is the number of such pieces manufactured during an hour (hourly output) or a shift (shift output) with work properly organised.

For welding operators the standard time is referred to one metre of weld, and the standard output is the number of metres welded per day or shift.

The standard time is made up of the following items:

(a) preparation and closing time, which is calculated for a batch of pieces; it includes the time for taking out an assignment, briefing, setting up the necessary fixtures, acetylene generator and torch, and for turning in the work done;

(b) production time, which is calculated for one piece or one metre of weld and includes the time for welding proper and the heating of the piece in the beginning;

(c) auxiliary time which includes the time for welding rod replacement, flame adjustment in the course of welding, weld measurement and inspection, edge cleaning, removal of slag and sputter from the weld, setting-up and removal of the piece, marking, movement from one scene of operation to another, rest, etc.;

(d) servicing time, which includes the time for arranging and removing the tools, setting up and replacement of cylinders, connecting the hoses, servicing the generator, igniting the flame, adjusting and cooling the torch.

Naturally, the production time is the basic item of the standard time. It varies with the nature and thickness of the piece, the heat output of the torch, the welding technique employed, the welding position used, and the skill of the welding operator.

Before the standard time can be found, the production time should first be determined and an allowance of 10 to 40 per cent added to it to take care of the operations listed in (a) through

(d). This allowance is smaller when work is properly organised; as a result the productivity of the welding operator increases.

The production time, τ, for one metre of weld can be found with an accuracy sufficient for practical purposes from the following relation:

$$\tau = KS,$$

where S is the thickness of the metal in mm and K is the coefficient of proportionality, min-mm/metre, which depends on the nature of the metal to be welded.

The values of K are 4 or 5 for low-carbon steel, 6 for alloy steel, cast iron and copper alloys, 3.5 for copper, and 4 for aluminium.

The production time for welding low-carbon steel may also be taken from Table 65.

Table 65

Production Time for Welding Low-carbon Steel (in minutes)

Weld	Plate thickness, mm								
	0.5	1	2	3	4	5	6	8	10
Flange, closed, without filler metal	4	—	—	—	—	—	—	—	—
Butt, square, from one side	—	9	10	11	12	14	—	—	—
Butt, single-bevel, 70 deg. included angle .	—	—	—	—	—	20	25	30	50

When making vertical or overhead welds or welding other metals, the data in Table 65 must be multiplied by the following coefficients which allow some time for reduced weldability:

Brass and bronze . . .	1.2	Aluminium and its alloys	0.6
Cast iron	0.77		
Copper	0.85	Vertical welds	1.2
Alloy steel	0.70	Overhead welds	1.6

The consumption of the welding materials for average welding conditions (as a function of the metal thickness S in mm) may be taken as follows:

Acetylene	$8S$ litres per linear metre
Oxygen	$9.5S$ litres per linear metre
Filler metal	$10S$ grams per linear metre (for a bevel angle of 45 dergees).

ACCIDENT PREVENTION
IN FLAME WELDING AND CUTTING

1. General

Only authorised and competent persons, duly briefed in safety rules and qualified, may be allowed to operate welding equipment and to do welds.

Welding must never be done near inflammable and combustible materials (such as petrol, kerosene, tow, shavings, etc.). In welding in the open, the scene of operation should be at least within 10 metres of combustible materials.

When welding inside tanks, vessels, boilers or close spaces, the operator must be allowed to interrupt work regularly for a rest in the open. As an additional precaution, a helpmate must be present outside the vessel being welded. The lighting fixtures used in such cases should operate on not more than 12 V, which is not dangerous to life. There must be a fan set up outside the vessel to supply fresh air into and to withdraw welding fumes out of it.

Regular welding operations, other than field welding, must be performed in a closed room with ample ventilation. The welding room should be not less than 10 sq m in area with at least 4 sq m per welding station and passages not less than 0.8 m wide between the stations.

2. Calcium Carbide

The main objective of safety rules for handling calcium carbide is to prevent the formation of an explosive acetylene-air mixture. This mixture may be formed when water or water vapours act on calcium carbide, causing it to decompose partially. The air-acetylene mixture is explosive if it contains 2.2 to 81 per cent acetylene (by volume). This mixture may be detonated by a spark due to impact, friction, or short circuit, and by the temperature rising to above 450 or 500° C.

The temperature may rise to a point where the air-acetylene may explode when the carbide is decomposed with too little

water present or when too much sludge has accumulated. This is especially true of finely divided carbide and, particularly, carbide dust which decomposes at a high rate and, with too little water present, may give rise to local heating in the reaction zone to dangerous limits. In the presence of air, this may set fire to the exuding acetylene or cause the mixture to explode.

The air-acetylene mixture may be present in carbide drums. Cases are known in which lack of care in opening carbide drums resulted in casualties. This is why carbide drums should not be opened with tools which may produce sparks, i. e., instead of steel chisels and hammers, use should be made of brass tools.

Carbide drums should by no means be opened with an oxy-gas torch. It is likewise dangerous to smoke near a drum being opened or bring an open flame very close to it. When emptying a carbide drum, care must be taken not to let carbide lumps fall onto steel surfaces from a great height. The reason for this precaution is that commercial calcium carbide often contains lumps of ferro-silicon which is formed when carbide is melted in a furnace. Striking at one another or at a steel surface, these lumps may produce a spark. It is a good plan, therefore, to pick the ferro-silicon out of the carbide by hand or with the aid of a magnetic separator.

When a sample has to be taken from a carbide drum, the procedure should be as follows: pour some mineral oil in the middle of the drum head; drill a hole in the drum head with a hand brace; cut out a wedge around the hole with a pair of shears some 150 mm long and at an angle of 70 or 80°; pry up the wedge and take a sample through the opening thus formed.

Calcium carbide should never be stored in open drums. Instead, it should be either held in a can with a hermetically sealed lid, or the hole made in the drum head must be closed with a sand-filled bag.

Calcium carbide wetted with water (such as discharged from a defective acetylene generator) must never be stored in a confined space. It should be immediately cleaned of slaked lime and utilised in another generator or completely decomposed in a vessel holding enough water, while stirring it continuously.

Utmost care must be taken in handling carbide dust which decomposes almost instantaneously and is heated quickly, even if very little moisture (water vapour, drips, etc.) gains access to it. Empty drums should be carefully cleaned of all traces of carbide dust and stored in the open under a shed after they have been dried, aired and put up in stacks. Carbide dust should be stored in perfectly tight, closely sealed containers. If it is not

utilised for acetylene generation, carbide dust should be decomposed in the open air, using a vessel holding 800 litres of water or more. Dust should be charged into the vessel in small increments (200 or 250 grams), while stirring the water in the vessel vigorously. The water-to-carbide ratio should be not less than 10 litres per kilogram of carbide dust. Another charge of dust may be poured into the vessel only after the previous one has decomposed completely.

Carbide dust must never be slaked directly in sludge pits, as this may result in an explosion of the acetylene-air mixture in the pit.

3. Acetylene

As has been noted already, acetylene and air form an explosive mixture over a very broad range of ratios. Therefore, the primary consideration of safety in handling acetylene is to prevent the formation of explosive mixtures. Accordingly, every piece of acetylene generating or using equipment, piping and fittings shoud be absolutely tight at joints; every room where any amount of acetylene may escape to the surrounding atmosphere must have ample ventilation.

Whenever a smell of acetylene is felt, the room should be immediately aired. Smoking, use of open flames or spark-producing tools should be forbidden in spaces where acetylene may escape. Acetylene equipment should not be approached closer than 10 metres with an open flame, a cigarette or a smouldering material even in the open air.

It is essential that acetylene generators and piping be blown out at regular intervals, have the necessary excess of water, and cooled thoroughly where acetylene is liberated from calcium carbide.

Where gaseous acetylene is stored or transported in ordinary vessels or generators, the pressure should never be raised to above 1.5 atm (gauge), as otherwise acetylene becomes explosive.

When acetylene is compressed by means of gas blowers or compressors, it is essential to provide better cooling of the gas and to feed it to filling equipment over small-bore tubes (8 to 25 mm) in which the propagation of the shock wave is handicapped. Where a greater area of effective opening is necessary, use should be made of several tubes of small diameter rather than of a single large-diameter pipe.

Acetylene cylinders should use only tested and certified porous filler soaked with the specified amount of acetone.

Cylinders with the filler forming free spaces in them should not be allowed for use, as the acetylene in such a cylinder may explode.

Acetylene valves and pipe fittings should not use commercial copper or copper alloys containing over 70 per cent copper, as this may produce copper acetylide, an explosive compound. This compound tends to flash due to friction or impact and may, therefore, detonate the acetylene or the air-acetylene mixture in the cylinder or system.

The air-acetylene mixture may also explode where much compressed acetylene is leaking at a joint or through a broken acetylene tube. In such a case, the leaking gas may be ignited by friction, impact or spark, setting fire to the premises. A similar case occurred at the dissolved acetylene factory in Borzigwald, Germany, in 1929.

Water should never be used to put out a fire, especially where calcium carbide may be present. Instead, use must be made of inert gases, such as nitrogen or carbon dioxide, or of dry and CO_2 fire extinguishers, sand or felt.

In all cases, measures must be taken to shut off the supply of fresh air and acetylene to the fire and to disconnect the remaining pieces of acetylene equipment.

4. Acetylene Generators

As far as acetylene generators are concerned, the main point to be watched is the possibility of the calcium carbide and acetylene in the reaction zone being heated to a very high temperature. This condition is dangerous in that, should some air gain access into the gas-generating chamber, the resultant air-acetylene mixture may explode. Another source of accidents may be the escape of acetylene into the generator room, also producing an explosive air-acetylene mixture.

Every acetylene generator should be fitted with a certified water seal to prevent back fires to the generator from the torch. This is especially true of portable generators which are connected to their torches directly by a length of hose.

Acetylene generators should use calcium carbides in lumps of the specified size. A very grave danger is to use carbide dust or smalls in generators designed for operation on large-size lumps of carbide, especially in the "water-to-carbide" type. Failure to observe this rule may result in a very violent gas formation, an increase of pressure in the generator, the escape of acetylene into the atmosphere, and the heavy overheating of the carbide in the gas-generating chamber with all consequences.

For the same reason, an acetylene generator should never be run at the upper limit of its gas-generating capacity; nor should more gas be withdrawn from it than is specified in its certificate. It is dangerous to open the gas-generating chamber too early, especially when too much undecomposed carbide has been left and this carbide has been wetted with water. The danger lies in the fact that the air getting into the chamber makes an explosive mixture with the acetylene, and the hot carbide lumps may ignite it. The mixture may also be ignited by the phosphorus compounds present in the hot carbide lumps.

The presence of ferro-silicon in carbide is dangerous for the "carbide-to-water" type of generators, as in falling down lumps of ferro-silicon may strike at the steel parts of the generator and produce a spark. To eliminate this possibility, large acetylene generators sometimes have the inside surface of their charging units lined with rubber or duralumin.

After a generator has been charged with carbide, it should be blown out to remove all air from it and its piping. The purging gas should be discharged into the atmosphere over hoses or exhaust pipes.

In winter time, to prevent the water in the generator and water seal from freezing, the generator must be heat-insulated. Should all or part of the water in a generator freeze, it may be warmed up only with hot water or steam. Any other source of heat (an open flame, hot objects, etc.) is extremely dangerous to use as it may set off an explosion.

Whenever an acetylene generator which has been in service has to be repaired, it should first be purged of traces of acetylene with some inert gas (nitrogen or carbon dioxide), washed with water several times and cleaned of acetylene lime. In washing, the water should be fully changed at least three times, or traces of acetylene may be left in the generator.

An acetylene generator which has been in service may be welded for repair in the open air only and after it has been cleaned of all traces of acetylene and carbide lime as advised above.

In servicing acetylene generators only brass or silumin tools which do not produce sparks may be used.

In removing sludge from the gas-generating chamber of "carbide-to-water" generators, it should at the same time be filled with water or acetylene to avoid a vacuum building up in the chamber and air finding its way there. Carbide sludge must never be discharged into a severage system, as the acetylene liberated from the sludge may form an explosive air-acetylene mixture in the pipes.

Should acetylene take fire while a generator is being unloaded, the fire should be immediately put out with a dry fire extinguisher, sand or felt. If acetylene is liberated by water-moistened carbide, the latter should not be dumped into a sludge pit, as it may set off an explosion in the pit.

When the supply of water to a generator fails for some reason, no more gas should be withdrawn from or carbide loaded into it. The remaining carbide may be removed from the gas-generating chamber only after the generator has been fully cooled and the pressure in it has been brought down to the atmospheric pressure.

Generators should in all cases be serviced in compliance with the Manufacturer's instructions by competent and authorised personnel.

Stationary acetylene generators should be installed in well ventilated rooms with a volume of not less than 60 cu m and an ambient temperature of not below +5° C to prevent the water in the generator from freezing.

5. Water Seals or Flashback Arrestors

Only tested and certified water seals may be employed on acetylene generators and acetylene lines, if full protection against flashbacks is to be provided. Every water seal should be inspected at least twice a shift for water which must be level with the pet-cock. Too much water in a water seal is as undesirable as too little of it.

A water seal will operate trouble-free only when the actual pressure and rate of flow of acetylene do not exceed the values stamped on its nameplate. At pressures and flow rates higher than that the water seal may not be able to arrest a back fire.

It should be remembered that one water seal may serve only one welding or cutting torch.

Water seals should be inspected for the condition of their parts (gas distributors, seats, moisture eliminators, etc.) at least every month, especially after a back fire, as it may have damaged the seal.

A water seal must be mounted in a vertical position and have a cock on the pipe admitting acetylene to it. In filling the water seal with water, it should be disconnected from the acetylene line, and the pet-cock should be opened. When no acetylene is to be withdrawn from the associated generator for a long time, the cock on the gas pipe should be closed. In winter, when operating outdoors or in an unheated space, water seals must be emptied of their water after work, or it may freeze. Alternatively, water seals may be filled with an aqueous solution of sodium chloride or cal-

cium chloride. As much as 15 per cent sodium chloride in the solution reduces its freezing point to —10° C, while a content of 20 per cent calcium chloride does so to —20° C. Use may also be made of aqueous solutions of ethylene glycol and glycerine (anti-freeze agents).

6. Gas Cylinders

While in transit, gas cylinders must have their caps on to protect their valves from damage and dirt. Gas cylinders must never be transported without their safety caps on. In loading and unloading cylinders, filled and empty alike, care must be taken to guard them against shocks and falls. Cylinders must be set up at least 5 metres away from a welding or cutting torch. They must never be set up near furnaces, heating appliances or other sources of heat.

Each welding station may only have two cylinders: one in use and the other as a stand-by. Cylinders may be stored fastened to walls, columns or on special trolleys. In field welding, cylinders may be placed sideways, tilted a little on a wooden pad so that the valve is a bit higher than its shoe. In a shop having more than ten welding stations, preference should be given to the centralised supply of gases over pipelines.
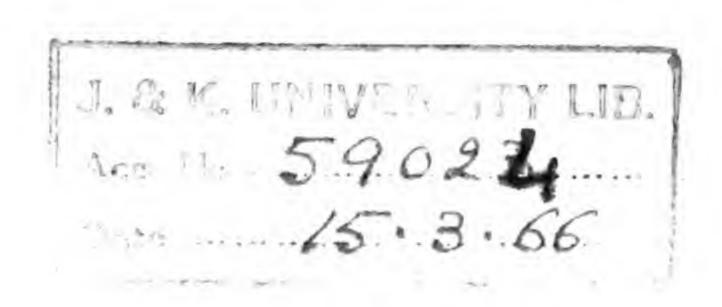
7. Pressure Regulators

Use may only be made of pressure regulators with good pressure gauges. Oxygen pressure regulators must be protected from ingress of oil and grease. Before connecting a pressure regulator to a cylinder valve, it is vital first to examine and blow out the valve. Oxygen should be admitted to a pressure regulator by gradually opening the cylinder valve with the adjusting spring of the regulator made fully slack. In admitting gas into a pressure regulator, the operator should take care not to stand in front of the regulator. Watch must also be kept on gas leakage in the regulator, at the joints between the regulator and valve and hoses. Should there be any leaks, they must be eliminated immediately.

8. Welding and Cutting Torches

Utmost care must be taken in handling welding or cutting torches. They must be protected from damage and dirt, watched for leakage at the joints, and repaired without delay. Before igniting a torch, the hose connecting the torch and its water seal must

first be blown out with acetylene to remove traces of air. In the case of injector-type torches, the acetylene passages must first be checked for sufficient suction, while the oxygen jet is allowed to pass through the injector. In turning off the torch either after operation or in the case of a back fire or flashback, the first to be closed is the acetylene cock and then the oxygen cock. At any moment during welding or cutting, the operator must not forget about the flame: it should be so directed as not to injure those standing near-by hoses, cylinders or any combustible material.

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* Only the most recent of the references given in the Russian edition are quoted. They are all available in Russian. — Ed.

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